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The Expiration of the Hall Patents.

On April 2, 1889, five patents were granted to Mr. Charles M. Hall for the production of aluminium, numbered 400,664, 400,665, 400,666, 400,667 and 400,766. All these patents relate to the electrolysis of alumina dissolved in fluorides. By their expiration on April 2 of this year the composition of the only electrolyte used in this country and abroad for the commercial production of aluminium has become public property. This does not affect, however, the commercial monopoly of the Pittsburg Reduction Co. for the production of aluminium in this country. While the electrolyte is public property, yet the method of heating the electrolyte by the electric current itself -which is unavoidable when working on a large commercial scale-will be protected by the Bradley patent till 1909. The outcome of the Bradley patent litigation, which was covered in detail on pages 510 and 560 of our first volume, now begins to become a blessing for the Pittsburg Reduction Co., since it has almost three more years to further strengthen its already extremely strong industrial position. The enormous increase in capacity of its works in recent years has only been sufficient to just meet the increasing demand. As a matter of fact, there is now almost a scarcity of aluminium in the market. While the boom of the automobile industry and the increased use of aluminium for electric conductors has much to do with this situation, yet it is certain that the better understanding of both the excellent qualities and the limitations of metallic aluminium has resulted in an increase of consumption on all lines, and that this increase will continue at about the same rate during the next years.

New Developments of the Fused Electrolytic Bath.

It is supposed that the chief trouble with the Salom lead reduction process was the smallness of the unit. This evidently has not been successfully overcome by the new continuous cell. The process was so attractive at first sight with the recovery of sulphur and the small amount of power used per ton of raw material, that the slowness of commercial development must be attributed to some cause, and this is the most likely one. Three different experimental metallurgists have endeavored to accomplish the same end as Salom had in view, but with the use of larger units by recourse to the fused electrolytic cell. In the Analysis of Current Electrochemical Patents in our April issue we noted a patent granted to Mr. Clinton Paul Townsend on this general subject of electrolytic lead reduction. In the Townsend electrolytic lead process the cathode employed is molten lead, on which rest the particles of galena. These particles are subjected to "cathodic reduction" while the sulphur is liberated at the anode. The action can also be described as the electrolysis of sodium sulphide formed in the cell. The sodium then reacts on the galena with the reformation of sodium sulphide. In either case the end

result is the same. Any impurities in the lead ore are removed "in any suitable manner." Such is the outline of the Townsend process which has much in it to be commended and in the recovery of sulphur as by-product. Two other similar processes are described in our current issue-the Betts-Valentine, in an article by Mr. Anson G. Betts, and the Swinburne-Ashcroft process, in an article by Mr. Edgar A. Ashcroft. In the Swinburne-Ashcroft process the decomposition of zinc sulphide in suspension in the fused electrolyte of mixed zinc chloride and sodium chloride is employed, and also the electrolysis of lead sulphide dissolved in a fused electrolyte of lead chloride. Mr. Ashcroft believes that, notwithstanding the cheapness of the old furnace process and also the "Scotch Hearth" process, that the Swinburne-Ashcroft process is commercially promising. We will comment on the very interesting work done by these two distinguished British electrometallurgists at greater length in a later issue, after the conclusion of Mr. Ashcroft's articles on his work with fused electrolytes.

. . . Mr. Betts has made extended and most interesting metallurgical experiments on analogous lines. He gives an exhaustive analysis of the future prospects of his processes in the article on electrolytic lead smelting. In the short course of a few years, Mr. Betts has developed, in spite of obstacles incident to any new process and in spite of the amused tolerance of older metallurgists, his electrolytic lead refining process to a successful commercial basis, where he has two plants of fairly large size in successful operation, while the third one is soon to be started. Therefore, anything that Mr. Betts publishes is worthy of careful consideration. The object of his work was to dissolve galena from a high-grade lead concentrate in a bath of an easily-fusible electrolyte. The electrolyte selected was lead chloride, and thus the process is similar to the Swinburne-Ashcroft process. It was thought at first the process would be commercial, because of the recovery of the sulphur as a byproduct, and also because of the fact that there would be practically no loss in lead. It was reckoned that this would give a saving of from \$7 to \$8 per ton over the old, tried smelting processes.

However, in spite of such a first favorable outlook the same old spoke in the wheel-"the accumulation of impurities in the electrolyte" (forcibly and clearly elucidated by Mr. David H. Browne in former issues of this journal) blocked this process, as it has blocked in the past many other promising electrolytic processes. Accordingly it was seen that some method must be devised to furnish the cell with a worked-up purer material. Just as in the Hall process, pure alumina must be made by a chemical process, so Betts' attempt to prepare a refined lead sulphide-practically a lead matte to feed to his electrolytic lead cell. There is an exact parallel between the aluminium process and Betts' proposed lead process. And again we have the evidence of the truth of the generalization that electrometallurgical methods are especially adaptable to the finishing reaction. Mr. Betts has deviated, however, somewhat from this general line of work, and has determined to make matte by electric smelting, and thus to eliminate the impurities by a smelting process. He also proposes to combine the preheating of his charge by coal in a reverberatory furnace and finishing the matting in an electric furnace. Very careful thermochemical analyses are given of the two operations, and comparison is made with the costs by the old smelting process. An important by-product is, of course, sulphur, and the combined treatment has an additional advantage, dué to increased recovery. Considered all in all, the Betts process for electrolytic lead smelting is quite attractive, and although it will be undoubtedly the subject of much criticism by other engineers, nevertheless, it is only right to make proper allowance for the success of the inventor in other lines. There is a striking similarity in a general way between the proposals of Townsend, Swinburne, Ashcroft and Betts. All of their proposals are interesting and illuminating, in the light of future possibilities, and also in showing striking analogies to other processes. We hope to publish in the course of time additional matter in this new field of electrochemical development.

The Electric Furnace for Steel Refining and Iron Reduction.

We are authoritatively informed that after some unavoidable delay in equipping the plant, the Héroult electric steel refining process is now finally in successful operation at the works of the Halcomb Steel Co., in Syracuse. The electric furnace was first started on April 4, with cold charging materials; a week later the regular operation began, superoxidized steel being supplied in molten state from a Wellman furnace to the electric furnace for refining. From the very start the operation has been satisfactory in every respect. There has been no hitch whatever, so that the success of this furnace, which is the first large electric steel furnace in this country, is complete. We offer our hearty congratulations to both Mr. Halcomb and Dr. Héroult.

. . In our last issue we gave an account of the successful outcome of the experiments made by Dr. Héroult at Sault Ste. Marie, with the cooperation and under the supervision of the Canadian Government, represented by Dr. Haanel. These experiments related to an entirely different metallurgical problem, namely, the reduction of iron from ore. This problem is, of course, a much bigger one and of vastly greater commercial importance than steel refining. On the other hand, it is certain-and we have repeatedly discussed the reasons why-that the electric furnace will be of very much less commercial importance for the problem of iron reduction from ores than in the steel industry. It can no longer be doubted that for refining steel, for making all kinds of high-grade steels, to compete with crucible steel, the electric furnace is specially well adapted, and in this field it will reign supreme within a very few years. There also seems to be a bright future for a large electric steel mixer, to secure uniformity in the product of an entire works, as proposed by Dr. Héroult and described on page 30 of our January issue. Further, it is not impossible that a suggestion made by Dr. J. W. Richards, in his address to the American Electrochemical Society two years ago, will be realized in practice; namely, the use of electric heat as an auxiliary with our open-hearth steel furnaces to take off the "peak of the load," the electric current furnishing the last few hundred degrees of necessary temperature, while the combustion of gas furnishes the entire lower range.

As a matter of fact this same combination of gas and electric heating has recently been introduced on a fairly large scale with great success in an entirely different chemical industry in this country. On the other hand, it is certain that for the reduction of iron from ore the electric furnace can successfully compete with the blast furnace only in special localities under conditions which are especially favorable to the electric process and unfavorable to the blast furnace. These conditions are essentially cheap electric power and scarcity of coke. Under the conditions existing in Canada it is quite possible that localities may be found where these requirements are fulfilled.

Several points brought out in the Soo experiments are noteworthy, since they prove certain distinct advantages of the electric furnace process which should carefully be taken into consideration. First, there is no difficulty in using in the electric furnace ores of comparatively high sulphur content for the production of pig iron containing only a few thousandths of 1 per cent of sulphur. Further, charcoal and peat coke may be substituted for coke as the reducing agent. Finally, there is a possibility of utilizing to the utmost the calorific value of the fuel in a way impossible in the blast furnace. This possibility is indicated in the furnace construction of Héroult, described and illustrated on page 152 of our last issue. The fundamental idea is this: The gases leaving the blast furnace contain a considerable amount of carbon monoxide, and are now utilized outside of the blast furnace, most effectively in gas engines. The reason why carbon monoxide is necessarily contained in the blast furnace gases is to be found in the reaction between carbon dioxide and carbon, yielding carbon monoxide. As long as the fuel is mixed with the ore there is, therefore, no possibility whatever to avoid the formation of the monoxide, since the rising gases get into contact with elementary carbon. This is the reason why Héroult does not feed the reducing carbon together with the ore into the furnace, but feeds it through a hollow central electrode, so that it is introduced into the furnace at the base of the charge of ore which is not mixed with carbon. A mixture of carbon monoxide and carbon dioxide is then formed at the base of the furnace, and as this mixture of gases passes upward through the charge there is a reduction of iron ore with a simultaneous conversion of part of the carbon monoxide into dioxide. But the reducing power of the gas mixture diminishes, the higher the gases rise, and it would be impossible to change all monoxide into dioxide by this reaction alone. At a certain height, Héroult, therefore, blows air into the furnace, and thereby burns the remaining carbon monoxide to dioxide. The higher the point where the air is blown in, the greater the quantity of monoxide consumed for ore reduction, and, therefore, the smaller the heat produced by burning the monoxide. On the other hand, the lower the point of air blast, the smaller the ore reduction and the greater the heat developed. It is evident that in this ingenious way Dr. Héroult is enabled to get the full calorific value out of the fuel in the furnace itself. From a chemical and metallurgical point of view the method is undoubtedly sound, although it may be necessary to overcome some mechanical difficulties in practice. In the main experiments at Sault Ste. Marie the

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air blast was not used, although two short runs were made with air blast. It is unnecessary to emphasize here the entirely different character of the air blast in the Héroult electric furnace and in the blast furnace. The subject is extremely interesting, and further developments will be carefully watched by all progressive metallurgists. But, as we said before, the big commercial developments will come for the electric furnace in the near future in this country not in the pig iron field but in the steel industry. The great success of the plucky enterprise of the Halcomb Steel Co. is bound to act as an encouragement to others.

The Internal Energy of Chemical Elements.

In thermochemical data we have always to do with the difference between two chemical energies. Thus the formation heat of HCl is the difference between the chemical energy of HCl and the chemical energies of H and Cl. For all practical purposes it is permissible to assume the chemical energies of the elements, in this case of H and Cl, to be zero. This is, of course, arbitrary, but is just as permissible as the assumption of any zero point in any other scale, and no error can arise therefrom as long as the elements themselves remain intact. In other words, as long as we do not know the absolute value of the internal energy of an atom, we may arbitrarily assume it to be zero. But the situation is different when we consider an atom to be a system of electrons, and take into account the possibility of its disintegration, as evidenced by the phenomena of radio-activity, if, according to Rutherford and Soddy's theory, radio-activity is a transmutation of an element into another element. The phenomena of radio-activity must, therefore, enable us to get at the value of the internal energy stored in chemical elements. We know now that these values are enormous, of an entirely different order of magnitude than the ordinary values of thermochemistry.

The practical impossibility of changing one element into another one thus appears in a new light. The change is practically impossible, because it would involve enormous quantities of energy, and we have not the necessary forces at our disposal. The old alchemist wanted to change cheap elements into more expensive ones; he simply looked at the matter. We know now that if such transmutation of elements was possible and spontaneous, it would pay on account of the energy evolved. Some figures from a recent paper of Soddy are suggestive. Let us assume that the element uranium, for example, which disintegrates to the extent of a thousandmillionth part annually, could be forced to disintegrate completely in the course of a year. From I gram of the element more than 1,000,000,000 calories could be evolved, and if it was possible to change this energy, not into heat but directly into electrical energy, this would be equivalent to more than 1,000 kw-hours, and would suffice to keep a 32-c. p. lamp burning continuously throughout the year. By the expenditure of about 1 ton yearly of uranium, costing less than \$5,000, more energy would be derived than is supplied by all the electric supply stations of London put together. But the trouble is that with the forces at our disposal we cannot control the rate of disintegration of radio-active elements.

Ithaca Meeting of the American Electrochemical Society.

The programme of the ninth general meeting of the American Electrochemical Society, to be held at Ithaca, N. Y., from May 1 to 3, is as follows:

Headquarters for registry and information will be at the Chemical Laboratory of Cornell University, room 33, where members and guests will please register and receive their badges as soon after their arrival as possible.

All meetings will be held in the Chemical Laboratory, Morse Hall, which is at the northwest corner of the University

The mornings will be devoted to the reading and discussion of papers, and the afternoons and evenings to visits and social meetings.

TUESDAY.

On Tuesday morning, May 1, at 9.00 A. M., a meeting of the Board of Directors will be held at the house of Prof. W. D. Bancroft, 7 East Avenue.

The first session will take place on Tuesday morning, May 1, 10.00 A. M., in the Chemical Laboratory. After an address of welcome by Dr. Schurman, president of Cornell University, the annual business meeting of the Society will be held, at which the reports of Officers and Directors will be presented. The following papers will then be read and discussed:

W. H. Walker.-"An Instructive Laboratory Experiment in Applied Electrochemistry.'

Lamar Lyndon.—"Electrolyte Density in Storage Cells."

C. L. Collens, 2d.—"Some Principles of the Resistor Furnace

A. Van Winkle.—"Electrogalvanizing."

H. M. Goodwin and R. D. Mailey .- "Physical Properties of Fused Magnesium Oxide."

F. F. Colcord.—"Impurities in Electrolytic Copper."

G. I. Kemmerer.—"Cathodic Disintegration of Carbon in a Fused Chloride Electrolyte."

F. K. Cameron.-"The Movement of Suspended Clay and Other Similar Particles Under the Influence of the Current."

O. P. Watts.-"New Silicide of Molybdenum."

O. W. Brown.-"The Reduction of Metal Sulphides."

G. R. White.—"Laboratory Resistance Furnaces."

Tuesday afternoon will be devoted to an inspection of chemical laboratory, dynamo laboratory, engineering laboratories and shops.

On Tuesday evening, 8.30 P. M., the retiring president, Prof. W. D. Bancroft, will present in the Chemical Laboratory his address on "The Electrochemistry of Chemistry." This is to be followed by a complimentary smoker at the Town and Gown Club, Stewart Avenue, at 9.45 P. M.

WEDNESDAY.

On Wednesday morning a session will be held at 9.30 in the Chemical Laboratory, where the following papers will be read and discussed:

E. A. Sperry.—"Electrochemical Processes as Station Load Equalizers."

W. C. Arsem.—"The Electric Vacuum Furnace."

B. E. Curry.-"Electrolytic Corrosion of Copper-Tin Alloys.

Maximilian Toch.—"Electrolytic Corrosion of Structural

C. F. Burgess and S. G. Engle.-"Observations on the Corrosion of Iron by Acids."

Isaac Adams.-"The Development of the Nickelplating Industry."

R. C. Snowden.—"Electrolytic Precipitation of Lead from Acetate Solutions."

C. F. Burgess and O. P. Watts,-"A Microscopic Study of Electrodeposits."

G. A. Hulett,-"The Cadmium Standard Cell."

M. LeBlanc.—"Electrolytic Chromium."

B. E. Curry.—"Electrodeposition of Bronze."

G. R. White.—"Ferromanganese Anodes in Caustic Potash." On Wednesday afternoon a visit will be paid to the filtration plant, the hydraulic laboratory, the power plant in the gorge

and the Ithaca Gun Co. The excursion will start at 2.30 from the Chemical Laboratory.

On the afternoon of Wednesday a subscription dinner will be served at 7.00 P. M. in the Dutch Kitchen.

THURSDAY.

The third session will be held on Thursday, at 9.30 A. M., in the chemical laboratory, when the following papers will be read and discussed:

M. deK. Thompson, Jr .- "The Free Energy of Some Halogen and Oxygen Compounds, Computed from the Results of Potential Measurements.'

R. von Foregger and G. F. Brindley.-"Report on Experiments with Fused Sodium Peroxide for the Regeneration of Air in Submarines."

W. D. Bancroft.-"Lecture-room Switchboard."

G. R. White.- "Alternating-Current Electrolysis with Cadmium Electrodes."

J. W. Richards.-"The Electrolysis of Caustic Soda."

E. A. Ashcroft.—"A New Sodium Process."

H. M. Goodwin and H. A. Wentworth.-"Electrometric Measurements with Reference to the Dissociation of Fused

L. Kahlenberg and A. S. McDaniel.—"Differences of Potential Between Manganese and Lead Peroxides and Various Aqueous and Non-Aqueous Solutions."

Henry S. Carhart and F. W. Willard.-"A New Electrolyte for the Silver Coulometer."

E. S. Shepherd.—"Errors in Pyrometry."

John Nelson.-"The Effect of Oxides on the Adhesiveness of Electrodeposited Metals.'

On Thursday afternoon an excursion will be made to the Remington Salt Co., with a trolley ride around the Loop and the Circle. The excursion will start at 2.30 from the Ithaca Hotel. The hotel headquarters will be at the Ithaca Hotel, State Street.

Ithaca may be reached either by the Lehigh Valley or by the Lackawanna Railroads.

The Regeneration of Air by Means of Fused Sodium Peroxide.

At a meeting of the Western New York Section of the American Chemical Society, held at Niagara Falls, April 3, Messrs. Brindley, FitzGerald and Bennie gave a demonstration of the regeneration of air by means of fused sodium peroxide. Mr. FitzGerald said in part as follows:

"It was originally intended that Dr. R. von Foregger should read a paper to-night; but as he, unfortunately, cannot be

present he has asked me to take his place.

"When sodium peroxide is caused to react with water under suitable conditions it is decomposed, forming a solution of caustic soda and generating oxygen gas. Unfortunately, sodium peroxide in its ordinary form is not a very safe substance to handle; but under the name of 'oxone' a new and perfectly safe form of sodium peroxide can now be obtained. This substance is manufactured by the Niagara Electrochemical Co.,1 the difficult problem of its production having been worked out by Mr. G. F. Brindley.

"In view of the action of water on 'oxone' it seemed reasonable to suppose that where moist air passed over it oxygen would be generated, and the caustic soda simultaneously produced would absorb any carbon dioxide contained in the air.

"Working on this hypothesis, Messrs. G. F. Brindley and

¹ Foersterling & Philipp, U. S. patent No. 788,256.

R. von Foregger carried out a number of experiments, and to-night we have one of these in progress before you. The box you see here is 150 x 70 x 70 cms. inside. It is tin-lined, and the lid provided with water-lute, which makes it perfectly air tight. The interior of the box is divided into two compartments by means of a wire screen. In one compartment is a wire cage, containing 'oxone' and a motor-driven fan, which causes the air to circulate over the 'oxone.' In the other compartment we have five rabbits. The lid of the box is provided with a tube passing to the interior, so that Mr. Brindley can draw off samples of the gas in the box and analyze them.

"We shall now stop the fan in the box, the experiment having been running, as you may see by the curve, for I hour and 50 minutes, and presently we shall observe a rapid increase of CO₂ and decrease of O. You will observe at the first part of the experiment how the oxygen percentage dropped and carbon dioxide increased before the fan was started.

"We believe that this experiment demonstrates conclusively that it is possible to regenerate the air in closed spaces by means of 'oxone,' and, moreover, the amount required is not large. Messrs. Brindley and von Foregger have found by experiment that one man requires very nearly 1-3 pound of 'oxone' per hour to supply him with oxygen for respiration and to absorb the carbon dioxide he eliminates. One other point in connection with the use of 'oxone' for the regeneration of air should be noted. You are aware that in the process of respiration various toxic substances are thrown off from the lungs, and that even where the oxygen and carbon dioxide are kept at a normal value in the air, the gradual accumulation of these toxic substances might have serious effects. experiments of Messrs. Brindley and von Foreger indicate, however, that one of the results of passing the air over the 'oxone' is to cause the elimination of these toxic substances."

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A short discussion followed Mr. FitzGerald's paper.

Electrometallurgy of Iron and Steel Before the Faraday Society.

The electrometallurgy of iron and steel was again the subject of discussion at the meeting of the Faraday Society on April 10. Three papers relating to this subject were presented, two being of French, the third of Italian origin.

The paper by Ernesto Stassano is a brief note on his rotating electric steel furnace in the artillery construction works of the Italian War Office, in Turin, Italy. The construction was described and illustrated in our Vol. III., p. 391. Three such furnaces, of 1,000, 200 and 100 hp. respectively, are being installed. They will be used for refining pig iron and melting scrap. But it is also intended to demonstrate the commercial possibility of making iron and steel in one operation direct from the ore.

The paper by Gustave Gin first states very briefly that the author's "canal-type furnace" (our Vol. II., p. 20) is now installed at the Plettenberg Works, Westphalia. These works were founded by M. Bruninghaus, who is associated with the Deutsche Elektrische Stahlwerke. Four photographs of the works are reproduced.

The author then introduces a description of his "furnace with canals and chambers," by the following words: "In the furnaces in which the canal is bent upon itself, the narrow cross-section of the canal is not convenient for the introduction of ore or scrap. Moreover, heavy currents at low potentials give rise to induction and other effects. To avoid these drawbacks I cause the electrothermic heating and refining to take place separately, the refining taking place in chambers which communicate with each other and with the leads by the canals for heating." However, "a difficulty which at present prevents the practical application of this furnace is finding a material

for the bases sufficiently solid to prevent short circuits between the branches of the canals and the chambers."

The author then takes up his "combination furnace," which consists of three chambers: First, a melting and oxidizing crucible; second, a deoxidizing and recarburetting compartment; third, a final mixing chamber. This furnace is essentially the same as the type described in our Vol. III., p. 372. The author makes, finally, some general remarks on induction furnaces, the original invention of this type being ascribed to Ferranti (English patent of 1885).

The author of the third paper, which is the most interesting of the three, is Ch. A. Keller, engineer to the Keller-Leleux Electrothermic Co., of Livet, France. He states that J. Holtzer & Co., of Unieux (Loire), have installed an electric steel plant which has just been completed. A steam-driven 1,500-hp. Westinghouse alternator gives a current of 20,000 amps., to be used in one Keller furnace. (According to the Canadian Commission Report, our Vol. II., p. 482, this is in principle identical with the Héroult furnace, which was repeatedly described in our columns, and varies only in details of construction.) The current is only for the finishing, refining reaction, since the steel produced in an open-hearth furnace is "run into the electric furnace immediately after the oxidizing melt, where the remaining operations would be completed by the current; 800 kilograms of metal could be operated on at one time, the operation being repeated three or four times in 24 hours."

The furnace at Unieux weighs about 50,000 kg.; it rests on a steel cradle and can be tilted. All the mechanical and electrical controls are obtained by auxiliary hydraulic motors. The steel produced by this furnace is intended for cannon and projectiles. No further metallurgical or mechanical information is given.

The author finally gives a preliminary account on experiments made by him at Livet on pig iron reduction in the electric furnace. "Apart from the elimination of sulphur, these experiments have shown that the Keller furnace (our Vol. II., pp. 483, 485) has an advantage over the blast furnace in the easier variation of the silicon and carbon content. This particular property of electric reduction would prove of great utility in the production of soft castings specially for mill work." The author has set up at Livet an electric furnace of 2,000 hp., which will be capable of producing about 20 tons of castings in 24 hours. It will be used specially for the industrial working of the process. The current strength will be 25,000 amps.

Recent Practice in the Treatment of Tin-Tungsten-Copper Ores.

At a meeting at the end of last year the Institution of Mining and Metallurgy had before them an interesting paper by Mr. F. Dietzch, on the "Treatment of Tin-Tungsten-Copper Ores at the Clitters United Mines."

The paper began with the explanation that large quantities of tungsten and tin had been picked from the waste heaps of the old workings of this abandoned mine, and it was therefore decided to reopen it with labor-saving rnachinery and the best appliances for separating the respective ores. It was initially proposed to work the mill as two separate systems, the one for the treatment of tin ores containing little tungsten, the other for treating tin ores containing large quantities of tungsten, but owing to the successful working of the magnetic separators, the two systems have been advantageously merged into one.

The arrangement of the mill is now as follows. The greater part of the ore arrives at the back of the mill and passes over two grizzleys, whose bars are 2 inches apart, the fines falling directly into the large storage hopper, while the coarse pieces fall on the platform, and from thence are fed into a 15-in. by

Io-in. Blake rock breaker and delivered also into the same storage hopper. The ore is then passed by four Challenge feeders into the boxes of four batteries, each of five stamps. The weight of each head is 800 pounds, and they are dropped ninety-five times per minute.

The pulp is passed through No. 25-mesh gunmetal woven-

A smaller portion of the ore is dumped into a storage hopper, and is, after the elimination of its fines by the automatic shaking screen, fed into a small 12-inch by 8-inch rock breaker. The ore then passes through a set of friction-driven rolls of 28-inch diameter, 12-inch face, and hoisted by the elevator to the vibro screen, from whence the fines are sent to a No. 6 ball mill, which crushes them through No. 30-mesh screening.

The coarse material collecting on top of the vibro screen returns to the rolls and is recrushed.

Prior to the introduction of magnetic separation, the second or smaller system was used for the treatment of tungsten-tin ores, and the place of the No. 6 ball mill was occupied by jugs, which received their pulp from the vibro screen after passing through a series of trommels. Following the extraction of the tungsten on the jigs, the tailings were recrushed in a separate five-stamp battery of the same construction as the other four above described. This separate battery is now supplied with mixed ores from the small rock breaker without having been crushed in the rolls.

The amalgamation of the two systems has permitted the plant to be better utilized, rendering the jigs superfluous in the general scheme. They will now be in reserve when re-elected, to be used upon such high-grade copper ores as contain the mineral in a sufficiently coarse condition to permit of their treatment by jigging. The rolls have proved of much use as an auxiliary to the No. 6 Krupp ball mill, as they increase the capacity of that mill considerably. The ball mill was experimentally erected to act as an auxiliary to the stamps, when it was found that although they could deal with the softer ores from Clitters, they were not of sufficient capacity when the harder ores of the Hingston Mine came to be included in the general treatment.

The combined pulp from the twenty-five stamps and the ball mill pass through three spitzlutten; two of these have three compartments of varying depths, and one has only two compartments. The product of each spitzlutten passes to a Buss swinging table, while the overflow of the three spitzlutten (fines) goes to a ten-compartment condensing and classifying spitzkasten, 50 feet long by 6 feet wide on top, and 5 feet 6 inches deep. Three pairs of these spitzkasten compartments supply their pulp through launders to three two-compartment distributing boxes, which in turn feed three double Lührig vanners.

The remaining four spitzkasten compartments supply a double vanner through a single two-compartment distributing box.

The overflow at the end of the series of spitzkasten, together with the tailings from the Buss tables, the overflow of the distributing boxes and the tailings of the Lührig vanners outside of the mill, pass into a second eight-compartment condensing spitzkasten, 40 feet long, 6 feet wide on top, and 5 feet 6 inches deep.

In future it is intended to collect the middlings from the tables and to recrush them in a wet ball mill, that is at present being used to recrush a stock of high-grade middlings, which had accumulated in the past. After recrushing, the table middlings, together with the overflow of the spitzlutten, will be treated on the Lührig vanners. At the same time, by using a No. 20-mesh screen, the first crushing of the ore will be made much coarser, and so decrease the production of slimes and increase the tonnage treated. It may here be remarked that the ball mill has been found to slime very much less than

the stamp mill, and that even after recrushing in a ball mill, through an 80-mesh, the production of slimes was found to be comparatively small.

It follows that while the tables finally eliminated a very large proportion of the light waste in one operation, and only allow waste particles carrying some minerals to be caught as middlings, the quantity of these latter is by no means large, and can be easily dealt with in a small ball mill, so that as a net result not only is the advantage of not sliming the ore attained in the first instance, but the capacity of the mill is greatly increased by this form of graduated crushing.

On account of the limitations of space in our present issue, the conclusion of the abstract of this paper, as well as the account of the animated discussion which followed, must be reserved for our next issue.

CORRESPONDENCE.

Construction of Lampboard.

To the Editor of Electrochemical and Metallurgical Industry:

SIR:—A lampboard was recently devised and made for me by Mr. Angel, which is so simple and inexpensive, and at the same time so effective, that I think a description of it may be of interest to some of your readers.

The arrangement consists of a rectangular wooden frame, about 70 cm. long and 7½ cm. wide, supported at either end about 15 cm. above the work bench.

On the two longer sides of this frame are stretched pieces of copper wire (BB) about 3 mm. in diameter, and connected to the binding posts (AA).

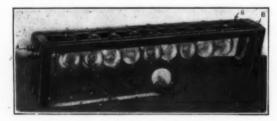


FIG. I .- CONSTRUCTION OF LAMPBOARD.

The lamps to be used are supplied with two small pieces of hard copper wire (EE) 5 cm. long, soldered to their terminals, as shown in Fig. 1. By means of these wires the lamps are hung across the leads BB.

A second and smaller rectangular frame, pivotted at C and

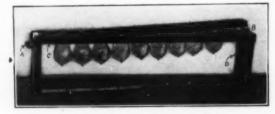


FIG. 2.—ONLY ONE LAMP IN CIRCUIT.

supplied with the small rack D, allows of any number of the lamps being thrown in or out of action. By raising the free end of this inner frame all the lamps may be successively cut out.

Fig. 2 shows the lampboard with all but the lamp nearest to $\cal C$ out of action.

LONDON, ENGLAND.

Electric Lead Smelting.

By Anson G. Betts.

There are possibilities for electric lead smelting, especially for roasted lead concentrates, using electricity as a mere heating agent in place of coke in the ordinary lead reduction, as the greater cost for power than that for fuel in the blast furnace process would be offset by a lower loss of lead. Lead is a volatile metal, and lead sulphide is also quite volatile, so any process, the blast furnace process for example, in which large volumes of gas or air come in contact with heated lead, will suffer from loss of lead in fume. The electric furnace, by reason of the absence of large currents of gases, is bound to give a higher saving of lead.

But this paper will be confined to what might be termed electrolytic lead smelting. Not a great deal has been attempted in electrolytic lead smelting, except by Swinburne, and possibly Swinburne and Ashcroft's well-known chlorine smelting process might be included under this head.

Dr. Wm. Valentine and I worked out the process which I shall describe, and it has since come to my knowledge that at least two other investigators* have made, or are making experiments similar to ours. In view of the fact that the process may be found to be very valuable under certain conditions, I think a presentation of our results and some calculations will be of interest.

It would not be a good plan to try to electrolyze lead sulphide directly for lead and sulphur, as the melting point is high, and lead sulphide is probably not an electrolyte, although a good conductor.

ELECTROLYSIS OF LEAD SULPHIDE DISSOLVED IN FUSED LEAD

The idea occurred to dissolve the lead sulphide in a bath, to form an easily melting electrolyte. A number of experiments were made, and for a time it appeared that we had a good process in the following:

A bath of lead was put in an iron or other suitable pot and melted lead chloride placed on top. A block of carbon or graphite dipped in the fused chloride as anode, while the lead was connected as cathode. Galena was added to the bath, and it dissolved rapidly, in large quantities, at temperatures well below a red heat. On passing a current, lead was reduced quantitatively, and sulphur came off from the anode, as vapor, if the bath reached the boiling point of sulphur, 440° C. The conductivity of the bath was extremely great. The voltage of decomposition is about .4 volt, but about I to 1.25 volts would be necessary to operate at a good rate of speed. The lead reduced was soft, free from iron and sulphur, the apparatus was durable, and with good galena, the results were excellent for a time. But eventually the cell became clogged up, on every occasion. It was evident that we had a good and neat process, if some way could be devised to make it continuous without too great expense. The power required to decompose a ton of lead sulphide would be about 14-hp. days, or a cost of \$0.77 or \$1.54 with power at \$20 and \$40 per horse-power per year respectively; 250 pounds of sulphur, or thereabout, would be saved, which would cover the power cost. The labor item, which would be for feeding ore to the pots and tapping out the lead and handling condensed sulphur, should not be over \$0.75 to \$1.00 per ton.

Then in cases where power was quite expensive, there would be the cost for keeping the pots hot with a fire underneath, which could be done most economically by producer gas, considering the resulting saving in labor and heat, but this would not cost to exceed 50 cents per ton. Allowing 50 cents for repairs, we would have a total cost of \$3 to \$4 per ton of galena, saving

practically all the sulphur, or 250 pounds per ton, and no loss in lead. These last two items would mean a saving over ordinary smelting practice of probably as much as \$5 to \$6 per ton. So that it was evident that the process had great possibilities.

To overcome the clogging of the pots we tried various plans, such as dissolving out lime with hydrochloric acid in a preliminary treatment, and trying to fish the accumulated silica, etc., out of the pots. These methods were not successful, and it was found that *iron* was the great nuisance as it is in all such processes for reducing lead. The iron would reduce with the lead and make a thick scum. The hoped-for reaction $Fe + Pb Cl_2 = Fe Cl_3 + Pb$ does not occur. It appears to cause serious loss of efficiency, too.

There seemed to be no way to get rid of the iron probably contained as pyrites, and taken with the expense caused by lime, silica, etc., the process had to be abandoned. It was found afterward that Swinburne had already described the identical process in a British patent.

Fig. 1 shows the type of apparatus that should be used. The pots would be made of cast iron, lined on the sides with fire-

brick, and could take a current of, say, 30,-000 to 50,000 amps., with a voltage of 1.25, or thereabouts, corresponding with a daily production per pot of 3 to 5 tons of lead.

Such currents could be produced probably more easily and cheaply than the smaller currents that have been used heretofore. With the com-

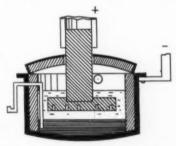


FIG. I.—APPARATUS FOR ELECTROLYSIS.

mutating generator the cost of the machine has been greater, the larger the current (for a given power output). With the unipolar generator, such as developed by Mr. Noeggerath, the cost of the machine for constant output should be actually less, the greater the volume of current. The machine described in the *Electrical World*, if built for a current of 7,200 amps. at 42 volts, instead of 600 amps. at 500 volts, would have been cheaper and simpler. It seems probable that in electrolytic furnace processes we will soon use much larger units for 50,000 or 100,000 amps., perhaps, produced by unipolar generators, and this will mean larger furnaces with consequent less heat loss, and less labor, hence higher economy.

Zinc chloride was also tried as electrolyte, but lead sulphide does not dissolve in it apparently. It was thought that zincy lead ores could be treated with this electrolyte, but zinc chloride suffers from many objections. It is relatively a much poorer conductor than lead chloride, too volatile and expensive to make.

We then worked out our process from the experience gained with the above process and other similar work, and made provision for the elimination of gangue and iron, which is necessary for continuous operation with commercial ores. The process consists of two stages; first, fusion of the ore into the matte, high in lead sulphide, to get rid of lime, alumina, silica, etc.; and, second, electrolytic reduction of the fused matte under such conditions that the iron remains fused as sulphide and does not contaminate the reduced lead or clog the bath. This condition is obtained in melted sodium chloride, is used as electrolyte, with lead sulphide resting on a cathode at the bottom of the cell.

It is not difficult to produce with lead ore a furnace charge which will melt easily into a matte and slag. Some of the ores that we worked on gave the following on analysis:

¹Engl. patent 10,329A, of 1897. ²U. S. patent of C. P. Townsend, 815,887, March 14, 1906; U. S. patent of Valentine and Betts, 816,764, April 3, 1906.

^{*}Electrical World and Engineer, Feb. 4, 1905, page 250; Transactions American Institute Electrical Engineers, January, 1905,

No.	Ore.	Pb.	S.	Fe.	Insol.	Zn.	CaO.	Cu	$_{\rm MgO}$
1	Payne, B. C	61.6	13.85	3.6	8.7	5-7	1.0	* *	* *
2	Missouri	72.5	13.3	1.45	1.25		4.65	.2	1.5
3	Bunker Hill and								
	Sullivan, Idaho	50.75	12.4	13-75	6.0		1.35		
4	St. Eugene, B.C	52.38	11.86	12.9	5.5	3.0	.8		8.8

No. 1 gave a good melt when fluxed with 15 per cent of limonite (carrying 51 per cent iron and 13.55 per cent insol.) and 2.5 per cent limestone. The products were slag and matte of the following composition:

	Pb	Fe.	Insol	Zn %	Fea %	CaO %	Wt. in Terms of Ore Used a
Matte	70.8	11.5	.8	5.1			78.7
Slag	12.12		44.7	6.7	23-4	3.3	31.7

In a good many of the melts metallic lead was produced in quite considerable quantity, which explains what would otherwise appear to be a heavy loss of lead. In this particular melt about 97 per cent of the lead is accounted for in the slag and matte.

No. 3 gave a good melt with 5 per cent of limestone. Percentage of lead which went into the slag 5. Matte recovered 68 per cent of weight of ore taken. Analysis of slag:

РЬ	SiO ₂	Feo	Ca	Percentage of Weight of Ore Taken.
5 pec ct.	46.20	22.6	8.33	22.75

No. 3 also melted without fluxes, but not so well. Matte recovered but not analyzed, 68 per cent of weight of ore taken. Analysis of slag:

No. 4 also melted without fluxes, giving 20 per cent of its weight of slag and 67.5 per cent of its weight of matte. Analysis of slag:

The mixture containing

melted well with limonite (51 per cent Fe, 13.6 per cent Insol.)
12 per cent, limestone 2 per cent, and coal .6 per cent. Percentage yield of matte 79.4 per cent, of slag 9.1 per cent. Analyses:

The coal was omitted in some melts, when no metallic lead was produced; 10 per cent of limonite, no lime used; yield of matte 86.4 per cent, of slag 16.2 per cent. Fusion not very successful. Analyses:

A great many such melts were made in large graphite pots, and we also tried melting in a small reverberatory furnace with gasoline burners. The latter, however, was not successful, on account of low temperature in the furnace.

For commercial work the melting could be done in blast furnaces, or better, in reverberatories, or still better, in an electric furnace. With an electric furnace there would be practically no loss of lead (the lead in the slag could be easily recovered by blast-furnace smelting), with a cost for power which can be calculated with a fair degree of accuracy. Taking the above melt with 12 per cent limonite, 2 per cent calcite, and 0.6 per cent coal, as an example, suppose the charge contains

																				Per Cent.
Zinc as	sulphide	0	0 0	 0	0	0	0	0	0	0	6			0	0	0	0	0		1.5
	st.lphide																			
Sulphur	*******												,	+		,		×	16	11.4
Iron as	pyrite																			8

Iron as siderite (in the lead ore)	. 3.3
Oxygen for same	8
CO2 for same	. 2.4
Iron as limonite	
Oxygen for same	. 2.2
H ₂ O	. 2.0
SiO ₂	. 4.4
Carbon	5
CaO as limestone	. 3.4
CO2 for same	. 2.6
	100.0

The calculation can best be made from the thermochemical values at the ordinary temperature, and adding to this the heat in the liquid products at 1,000° and the heat in the gases. The calculation was also made in an entirely different manner by taking the specific heats of the cold products, and adding the thermochemical values for the reactions that take place and the heat required for fusion. The result was not much different from the value obtained by the first method. The data used are from Dr. J. W. Richards' very useful "Metallurgical Calculations," published in the Electrochemical and Metallurgical Industry. Dr. Richards has very kindly supplied me with other data for use in the calculations. The heat in melted PbS at 1,015°, he has determined as 104 calories. The decomposition point of FeS2 and S he has also determined at 565°.

As nearly as it is possible to tell, from the work that has been done so far, the reactions, and heat values for them, for this particular charge, are:

Heat in Melted Matte and Slag. Calculation of the charge indicates that Fe₃O₄ was present in the matte, as it sometimes is, particularly in mattes high in iron. It hardly seems probable that there is magnetic oxide in these mattes, from the analyses given, and from other considerations. It is more probable that there is a slight error in the analyses.

Heat in melted matte at 1.000°:

$$\begin{array}{lll} \text{1.588 lbs.} & \left\{ \begin{array}{ll} 19\% \text{ FeS, ZnS, 200} \\ 5\% \text{ Fes,O_{1}, 400} \\ 76\% \text{ PbS, 104} \end{array} \right\} \\ \text{1.588} \times 139. \dots \\ \text{220,732} \end{array}$$

Heat in molten slag at 100° C.:

Heat in gases. Assume average temperature of evolution 900°:

209 lbs. CO ₂ , 209 × .289 × 900	5,850
For chemical reactions	360,183 96,115
Total	126 200

If the matte contained no Fe₃O₄ but FeS instead, the extra heat required for formation of FeS from pyrite, iron oxide and carbon is somewhat less than the difference in the heat in 33,420

the melted materials at 1,000°, so that there is no material thermal difference either way.

It will be seen that some reduction in the heat bill would come from the substitution of iron oxide for part of the lime, and especially so if the iron oxide was free from silica, as the amount of slag necessary to form would be less. In this case the limonite, added as flux, carried some 25 pounds of silica, and meant the production of about 75 pounds extra slag, increasing the heat consumption altogether by about 35,000 pounds calories, as follows:

	Lbs.	Cals.
Decomposition 27 lbs. CaCO ₃ by silica		7,200
Heat in 12 lbs. CO2 at 1,000°		3,720
Heat in 75 lbs. slag at 1,000°		22,500
Decomposition FeCO ₂ , unimportant	* * *	

As will be seen later the iron of the matte becomes in the process eventually available as flux without silica, and it is, therefore, proper to subtract from the above figure 35,000 calories, leaving 421,200 pounds calories.

Electric furnace efficiencies on a large scale may be brought to 75 per cent, especially when the temperature is relatively low, as in this case.

Therefore, we can estimate that 561,700 pounds calories are necessary to melt 1 ton of charge of the composition given, or 639,000 per ton of ore. Of course, this would be considerably less with some ores and charges, and greater with others, but an idea is at least obtained as to what is required.

One electric horse-power in 24 hours = 34,000 pounds calories, or 450-hp. hours per ton of ore. For 100 tons of ore per day about 1,880 hp. would be necessary. In some districts, with power at \$20 per year, for example, Niagara Falls, the power bill would be \$1.04 per ton of ore. With power at \$40 per year, for which it could be probably be developed in lead districts like Idaho and British Columbia, the power bill would be \$2.08 per ton.

It is easy to see how important economies of electric energy could be made for the above process. If carbon had been omitted from the charge, and a slag higher in lead made, the melting temperature would have been reduced, and less iron and lime needed for the charge. Allowing even considerable quantities of lead to go into the slag, would not be a disadvantage, as the proportion of slag produced is small, and there will always be enough lead in the slag to make it worth smelting in the blast furnace. The proportion of iron, and possibly zinc, in the matte is less, too, when carbon is not added to the charge, and this is a real advantage.

In case the ore is quite silicious, it might be better to flux the silica with lead, and resmelt the slag in a blast furnace, rather than add outside fluxes and increase the power consumption. If the ore is practically pure galena, like some Mississippi Valley ores, the amount of electricity required will not be much more than half the amount required in the example given.

For reverberatory melting, instead of electric furnace melting, and assuming a loss by volatilization of as low as 5 per cent of the lead, or 70 pounds per ton (valed at, say, 3.5 cents per pound = \$2.45 per ton), and remembering that the coal for the reverberatory furnace and its handling costs something, it is apparent that with reasonably cheap power the electric furnace has a decided advantage.

A combination process is also possible. The melting furnace could be built in an analogous manner to a lead hand roaster with fusion hearth. The ore could be warmed up to the oxidizing point by direct heat from a fire, and then pushed into the electric hearth at the end and fused.

Probably three-fourths of the direct heating and volatilization could be done non-electrically, and this would leave to be supplied electrically perhaps half. If power was very expensive this combination of process would pay. It would be cheaper to install, as the electric and power machinery to be supplied for this last of the work would be only half as much, while the extra brick work required in the furnace would be relatively cheap.

The simplest electric furnace for this work appears to be the type of furnace, illustrated in Fig. 2. Of course, other types in which the heat supplied cannot raise any portion of the charge to an unduly high temperature may be used.

The furnace consists of a fusion hearth, on the bottom of which rests fused matte, with, perhaps, some lead underneath

tit. Leading out from the hearth is a resistance channel running back again into the hearth on the opposite side. An alternating current is supplied by two large electrodes.

To produce motion of matte (or lead) through the channel, a small cross-channel is built in with two smaller electrodes at the ends. A magnetic field is established at the intersection of the channels. When a curis passed through the

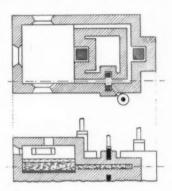


FIG. 2.—SECTIONS OF FURNACE.

cross-channel, motion is produced longitudinally in the main channel. Alternating current may be used, if an alternating current of the same frequency is also passed through the coil of the magnet.

The matte circulating through the channel becomes heated, and by establishing a continuous current the temperature on the hearth may be maintained. With this kind of furnace the consumption of electrodes in contact merely with metal or matte would be extremely small. The temperature at any one point would not be excessive, a desirable circumstance.

The labor involved in adding the ore to the electric furnace and tapping out the products ought to be a matter of perhaps 50 cents per ton. Repairs to a furnace of the type illustrated should be covered by 25 cents per ton. The cost for fluxes would vary largely, but as only 10 per cent need be used for many ores, and some of this is produced in the process, as will be seen further on, this would be an item of perhaps 25 cents per ton or less. The total cost would be for this operation, with power at \$20 and \$40 per year, as follows:

Labor	\$0.50	\$0.50
Repairs	0.25	0.25
Fluxes	0.25	. 0.25
Power	1.04	2.08
Smelting slag in blast furnace to per cent	0.40	0.40
	\$2.44	\$3.48

We still have to consider the electrolytic furnace operation for reducing the lead and recovering the sulphur. The theoretical voltage of decomposition of lead sulphide is .4 volts. This agrees with the value ascertained in the experiments. A somewhat higher voltage is necessary to drive the current through in the large volume desired to get a large output from the cells. External heating would save some electric power, but the more difficult construction of the pots, their larger size, and the greater amount of repairs, coupled with the fact that it would cost something to supply the heat by coal, puts such a procedure out of the question, unless power should be unduly costly.

The matte will be added at a higher temperature than the products are withdrawn, probably about 150° hotter, and this

will supply some heat to the electrolyzer. With larger furnaces and good heat insulation the voltage per vat should not exceed 2.4, and the current efficiency should be 90 per cent. Not all the lead may be reduced in the pot, as some will be tapped with the practically exhausted matte. Assume that the reduction will be 90 per cent of the lead. This figures out to 23.5-hp. days per ton of 75 per cent matte, or 20.7-hp. days per ton of ore. The 10 per cent of the lead tapped out with the matte will be brought back again as lead sulphide, so it will be better to add one-ninth to the amount of power, to save figuring the power for this lead at another time, so that the power per ton of ore would be 23.0-hp. days.

I think it quite possible that the voltage will be less than 2.4 volts per cell, especially with large cells.

Data: Hall's aluminium process is similar in some respects, and is operated in quite large cells, of 10,000 amps. capacity. The temperature is somewhat higher than in lead reduction, and the cells are lined with carbon, which is relatively a much better heat conductor than fire-brick, so the lead reduction ought to operate with considerably less heating by the current being necessary.

In the Engineering and Mining Journal, March 17, 1906, are given some data on the aluminium furnaces, as follows: Voltage per cells, 5.5; voltage required for electrolysis, 2.2; voltage for heating alumina and carbon, liquefying alumina and maintaining the temperature of the bath = 3.3.

To operate the 10,000-amp. cells, 14 pounds alumina and nearly the same weight of carbon per hour have to be added in the cold state.

Lbs	. Cals.
To heat 14 lbs, alumina to 900° , $14 - 90029$ To melt 14 lbs, alumina at 900° , 14×60 (assumed)	
To heat 14 lbs. carbon 900°, 14 \times .363 \times 900	
	0.068

Nine thousand and sixty-eight pound calories are equivalent to 0.48 kw-hours, or the e. m. f. supplied for heating materials and melting is 0.48 volts, leaving about 2.8 volts to maintain the temperature of the bath.

For a 10,000-amp, cell, reducing lead about 120 pounds of matte are added hourly, at about 150° above the temperature at which the products are removed.

Heat added hourly by matte and available for heating:

Ll	b. Cals.
With 103 lbs. PbS, $103 \times .12 \times 150$ With 14 lbs. FeS, $84 \times .21 \times 150$	1,854
	2,295
	Volts.
Equivalent to 1.2 kw-hours per hour, or,	0.12
Assumed voltage of operation	2.40
	2.52
Voltage of decomposition	0.04
Available for maintaining temperature	2.12
Plus 10 per cent loss of efficiency	0.04
	2.16

This is somewhat less than is supplied in aluminium making, but there are a number of considerations in favor of the lead process, namely, greater heat insulating power of fire-brick than carbon, possibility of using several times larger cells in both processes, while the current density may be higher, meaning less heat loss, because less surface. The carbons will not be detroyed in the lead process, so that it would pay to make permanent contacts and save something in power there.

Regarding the use of larger currents, in Richards' Alum-

inum, it is stated that the Hall process used about 9 volts per pot with a current of 1,800 amps. In the article in the Engineering and Mining Journal, above referred to, the voltage with 10,000 amps. is down to 5.5. There is no reason to suppose that with 30,000 amps., say, the voltage would be still lower.

In Acker's sodium process, which is similar in many respects, 6 volts per pot is stated to be all that is required when all unnecessary resistances are removed; 3.5 volts are required for decomposition and .59 volts to heat and melt the salt, leaving 1.9 volts to maintain the temperature. In Acker's process a current density of 2,750 amps. per square foot is used.

TA	DE	82	
1.43	DI.	J.	1.

Process	Current		Volts Available to Maintain Temperature								
Hall	8,000 to 8,200	6.0-6.5									

The reduction of the lead from the matte is carried out as follows: The fused (or solid) matte is put into a pot, as shown in Fig. 3, containing fused common salt. An iron casting imbedded in brickwork serves as cathode and container. It is lined internally with brick. The anodes are of carbon, much

better of graphite. The upper part of the carbons and the metallic connection can be protected by a sleeve of fire-clay filled with cement. At one side is a lead well, at the other side a sulphur outlet, and at the rear a tap for matte, and if desired to clean the pot

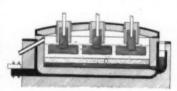


FIG. 3.—REDUCTION OF LEAD FROM MATTE.

for the salt electrolyte. There is no perceptible action on the graphite anodes. A mixture of salts for electrolyte is less desirable than straight NaCl, which we used almost entirely. With the mixture, one is apt to be troubled with a solid crust of one of the components of the bath on the anodes, stopping the current. This can be obviated by using a higher temperature, but then the advantage of a mixture of salts disappears.

With this kind of construction lead finds its way as far as the iron, through cracks in the brick-work, thus establishing the electrical connection to the bath of lead.

By using flues from a furnace built into the brickwork under the furnace, all loss of heat downward through the brick could be saved, and a lowering of voltage made, at the risk, however, of the pots cracking and lead running down into the flues. With this construction, if the pot leaks, no great harm is done. The greater part of the loss of heat will be laterally and upward, anyway. The pots could be heated by direct fire, and might last several months. It would be a case where experience would show for special conditions, which is cheapest, extra electricity, or new pots and cost of setting them.

It was discovered that the lead and iron sulphide of the matte do not dissolve in the electrolyte, so that the surface of the matte becomes cathode, and the action of the current on the start is to liberate chlorine at the anode, and reduce lead and form sodium sulphide at the matte cathode. After a short time sulphur chloride comes off, and then sulphur alone as sodium sulphide accumulates while the voltage drops.

The sodium sulphide formed does not easily dissolve in the salt electrolyte, but has considerable affinity for ferrous sulphide, and forms with it a fusible double sulphide. Yet there has been enough present in the electrolyte itself on long runs

⁶Page 383. ⁶Richards Am. Electrochemical Society, Vol. II., page 59. ⁸Trans. Am. Electrochemical Soc., Vol. I., page 171.

to continuously evolve sulphur free from chlorine at the anode. Nevertheless, the solidified electrolyte on one occasion after solidification contained merely a trace of Na₂S. Probably on cooling the Na₂S separated as the salt solidified.

The tapped matte contained in one case 8.4 per cent Na₂S and 8.5 per cent NaCl. The residue contained Pb 19.4 per cent, Zn 5.1 per cent, Fe 30.5 per cent, and Ca 2.8 per cent.

Calculating the residue as sulphides the composition of the

																															Per Cent
Na ₂ S						0		0																0 1					, ,		8.4
NaCl					0		0			0		0	0	0		0	0					0					, ,				8.5
PbS		0	۰	0					0				0	0	0						0		0								. 22.4
FeS			0	0	0						۰	۰		۰		n		0	0	0	0	0	٠	0	 			0 1		. 1	. 48.0
ZnS	0			0		0				0	0	0	0	0	0	0	0	0	0	0	D	0	0								7.6
CaS										0		0	0	0		0	0	0		0	0		-			. ,			0 1		. 5.0

On the assumption that the particular matte used contained 70 per cent lead and 9 per cent iron, which is not far from the truth, 48 parts of iron were added with 373 parts of lead, while 19.4 parts of lead remain, or a reduction of 96 per cent of the total lead. The remaining 4 per cent is not lost.

I mentioned that chlorine comes off with the sulphur if there is not sufficient sodium sulphide. As it is not desirable to produce chlorine with the sulphur it is necessary to return to the pots any sodium sulphide taken out with tapped matte, or replace it. This could be done by leaching the tapped matte and evaporating the solution for sodium sulphide. A consideration of the above figures shows that the amount of sulphide of sodium in the particular instance under discussion was 44 per cent of the weight of lead tapped, or only 22.6 pounds Na₂S per ton of fresh matte added. The amount is so small that the sodium sulphide could be produced directly by reducing salt-cake with carbon in a separate small furnace, to save leaching the matte tapped.

The matte tapped oxidizes very easily, and could be smelted in the blast furnace for the contained lead, along with the slag produced in the electric furnace, or better, it could be used as flux in the ore-melting process. The sodium sulphate and salt and iron oxide would be desirable fluxes.

It is not possible without actual large scale working to say just what the process would cost to operate, but it is possible to form a rough idea beforehand, and such a step is always a good one to take. It often saves laboring on processes which have no chance of success.

To add liquid matte to the furnace, handle the condensed sulphur, operate the pot, tap the lead and matte and dispose of them will require considerable labor. An estimate of 75 cents per ton of matte seems fair.

There will be no occasion for any great amount of repairs. We might roughly estimate repairs at 25 cents per ton of matte.

Twenty-five pounds of sodium sulphide, or to recover it from the matte tapped, might cost, say, 37.5 cents. Salt per ton matte, say, 8 cents.

matte, say, & cents.		
* **	Per Ton o	of Matte.
Power at \$20 per hpyear, 23-hp. days	\$2.54	\$1.27
Power at \$40 per hpyear, 23-hp. days		
Labor	-75	.75
Repairs	.25	.25
Sodium sulphide	.38	.38
Salt	.08	.08
Roasting tapped matte (no fuel necessary).	.25	.25
	\$4.25	\$2.98
Per ton of ore (1 ton ore = .9 ton matte)	3.83	2.68
Cost per ton for melting operation (see		
above)	3.48	2.44
Total cost	\$7.31	\$5.12
	Power at \$40 per hpyear, 23-hp. days Labor	Power at \$20 per hpyear, 23-hp. days\$2.54 Power at \$40 per hpyear, 23-hp. days Labor

2.15	cent 2.15	at 1	sulphur	pounds	215	for	Credit
\$2.07	\$5.16						

The figures are estimated to cover the bare smelting costs, and for comparison with mechanical or lime-roasting, followed by blast furnace smelting, the cost of the latter should be taken at not less than \$4.00 per ton, and often considerably more.

It is seen so far that the new process will have a chance in competition, but the lead loss has not yet been considered. Simple melting in an electric furnace ought to give no more lead loss than melting in crucibles, in which case there is practically no loss. There is no opportunity for loss in electrolyzing the matte. The only chances for loss are in roasting the "deleaded" matte and smelting the electric furnace slag. Remembering that there is only a small portion of the lead, say 4 to 10 per cent in the matte and 1 per cent in the slag, it is not probable that the lead loss from wet assay will be as great as 1 per cent.

On the other hand, a lead smelter will lose considerable lead, say 3.5 per cent loss from fire assay, or 5 per cent from wet assay, or more. That would mean for the electric process a saving of 60 pounds extra lead, worth in this country not less than \$2.40, and perhaps \$3.30. This should be charged to the blast furnace cost, making it in the United States, say \$7.20 or more.

There is another important consideration not to be lost sight of, and that is that the lead blast furnace requires other ores, and there are localities when no such ores can be had, and in that case the very important item of freight on the lead ore to a suitable locality must be taken into account.

Comparing the electric process with the blast furnace in a general way, the cost for labor ought not to be very different in the two processes, unless hand roasting is adopted, when the electric process has a decided advantage in this respect. With the electric process we have a cost for power of from \$2.18 to \$4.37, as against a cost for fuel and air blast in the roasting and reduction process of say, \$1.50 to \$2.00. The electric process will show no loss of lead when lead is determined by fire assay, and may come out a little ahead as against a loss of from 60-100 pounds lead per ton ore in the roasting and reduction process, and the electric process will also save about 200 pounds sulphur per ton of ore.

The roasting and reduction process can be operated successfully only where there is a plentiful supply of wet ores, while the electric process is independent in that respect.

I am very much indebted to Dr. J. W. Richards for much needed data and a number of valuable suggestions relative to the use of thermochemical and physical data for the ore melting process.

The Future of the Le Blanc and Electrolytic Alkali Works in Europe.

By John B. C. Kershaw.

An article by Hasenclever appeared in one of the German technical papers last year, discussing the future of the Le Blanc soda industry in that country. Hasenclever, who is the principal manufacturer by the Le Blanc process in Germany, considered the outlook for the industry in that country decidedly gloomy. Half the world's production of chloride of lime was stated by him to be already covered by the electrolytic alkali works. The ammonia soda works, which yield no by-products, were described as flourishing greatly, and are benefited rather than injured by the competition between the Le Blanc and electrolytic works.

The principal hope of the Le Blanc process, according to Hasenclever, lay in the discovery of new uses for sulphate of soda (one of the chief products), and in this direction the gradual growth of the new sulphur dye industry, in which

sodium sulphide was used, was expected to result in a large demand for sodium sulphate, the "salt cake" of the Le Blanc soda manufacture.

In view of the above pronouncement concerning the future of the Le Blanc alkali works in Germany, the reports of the chief companies carrying on the manufacture of alkali and chlorine products in the United Kingdom, for the year 1905, form interesting reading, especially for those who have a financial stake in the industry. The following are brief abstracts of the more important figures from the last reports of these companies:

The Castner Kellner Alkali Co. work the Castner electrolytic (mercury cathode) process at Weston Point, Cheshire, and their financial year ends on March 31. In their report for the year 1904-5 the directors state that the net profits' were £52,358, which, added to a carry over of £11,797, gave a balance of £64,135 available for distribution. Debenture interest and other charges absorbed £10,158 of this total, and the remaining £54,002 was divided as follows: Dividend, 4 per cent, £18,000; depreciation reserve fund, £15,000; plant patent and suspense account, £8,204; balance forwards, £ 12,798.

The Electrolytic Alkali Co. work the Hargreaves-Bird electrolytic process (diaphragm cell), at Middlewich, in Cheshire. Their financial year ends on Aug. 31. The report of this company for the year 1904-5 contains the following figures: Net profit, £4,746; carry over from 1903-4, £1,374. Total available for distribution, £6,120. Arrears of the dividend on the cumulative preference shares absorbed £5,601 of this amount, and £519 was carried forward to the next account.

The United Alkali Co. control the Le Blanc works in the United Kingdom, and their financial year ends on Dec. 31. The report of the directors of this company for the year 1905 has just been issued, and contains the following figures: Carry over from 1904, £43,762; net profit for 1905, £290,821; total, £334.584. This large sum is to be dealt with as follows:

Dividend on 7 per cent preference shares	£ 187,878
Transfer to reserve fund (depreciation account)	50,000
Transfer to reserve fund (general account)	40,000
Transfer to debenture redemption account	15,000
Carry forwards to 1906 account	41,706

£ 334,584

Brunner Mond & Co. operate the Solvay ammonia soda process at Winnington and Northwich, in Cheshire, and their financial year ends on March 31. The report of this company for the year 1904-5 showed a net profit of £512,532. A dividend of 35 per cent for the year on the ordinary shares was paid out of this huge balance, £100,000 was written off the book value of the leasehold mines in Wales, £2,500 was applied to the reduction of the patents account, and a balance of £88,735 was carried forward to the next year.

A study of the above reports, in conjunction with those that have preceded them, and some knowledge concerning the inner history of the industry, leads the present writer to take a less gloomy view of the future of the Le Blanc alkali works than Hasenclever. In the writer's opinion all three types of works (Le Blanc, electrolytic and ammonia soda) are likely to exist side by side for many years to come.

The character of the chief manufactured products may change, and some indications of this change are, in fact, already to be noticed in the output of the Le Blanc and electrolytic works. Bleaching powder and caustic or carbonate of the Le Blanc or electrolytic processes. It is to the by-products and special manufactures of each type of works, that one must look for the dividends of the future.

The following tabular statement, giving the average price of

soda will no longer provide large profits for their makers by

bleaching powder and 58 per cent soda ash in December of each year for the period 1895-1905, is of interest in this connection, since if studied in conjunction with the balance sheets of the various companies it serves to indicate the source of the profits of the three processes used in the manufacture of alkali and bleach.

Year. 1895	Bleaching Powder. £7.00	58% Alkali. £ 3.88	Year. Powder.	58% Alkali. £4.88
1896	6.63	3.50	1902 6.25	4.50
1897	6.13	3.88	1903 4.00	4.50
1898	4.65	4.00	1904 4.08	4.50
1899	6.00	4.38	1905 4.50	4.50
1900	6.88	4.63		

The writer's conclusions regarding the future of the alkalı and bleach industry generally may be summarized as follows:

The Le Blanc works will continue to produce caustic and carbonated alkali and bleaching powder, but in diminishing quantities, and the dividends earned by manufacture of these staple products (which once yielded large profits) will decline. But compensation for this loss will be found in the manufacture of sulphuric acid, sodium sulphate, hyposulphite, sulphide and other derivatives from these sulphur salts which cannot be manufactured by the processes of either of their rivals-and even the impurities of the pyrites burnt in the kilns for the lead chamber process of sulphuric acid manufacture, may be looked upon as an important source of revenue in years to come. Copper sulphate is another by-product, yielding its makers handsome profits, which can be most successfully made by the Le Blanc manufacturers. The Le Blanc Alkali Works will then depend more and more upon by-products, and less and less upon the staple products, soda and bleach, for their future profits. With this change will come the need for increased foresight and scientific supervision in their management and control.

The electrolytic works, when operating a good process, and when under wise financial management, will gradually take the place of the Le Blanc works as producers of bleaching powder, chlorates and other chlorine products. The manufacture of metallic sodium and of its derivatives, such as cyanides and peroxides, will also most probably fall into the hands of the electrolytic manufacturers. The cost of operating the electrolytic process of alkali manufacture is, however, greater than was at first estimated, and only the best processes and best equipped works will survive in the struggle for existence. The position of the electrolytic alkali industry in Europe at the present moment is of great interest in this connection, but limits of space will not permit any details relating to these works to be included in this article.

The ammonia soda works, both in this country and abroad, will continue to make enormous profits out of the manufacture of carbonate and bicarbonate of soda by the Solvay process. But, what is regarded by many as the best feature of these works, namely, the absence of by-products is, judged from another standpoint, a source of weakness. The ammonia soda works can never entirely take the place of the Le Blanc or electrolytic works, since neither sulphur nor chlorine products can be made a subsidiary and successful part of the manufacture of soda by the Solway process. All attempts hitherto made to extract chlorine successfully from the waste calcium chloride liquors of these works have failed, and the Hoepfner process of zinc extraction which has been in operation at Winnington for some years, cannot be considered to meet the demand for a satisfactory and profitable use for these waste liquors of the Solvay process. If such a process be ever found, and successfully developed, the forecasts given in this article will require revision, and their author may have to accept the fate of many another prophet, and to confess that in this world "nothing is certain but the unexpected."

LONDON, ENGLAND.

¹All values in this article are given in English pounds sterling-£

Laboratory of Applied Electrochemistry at Columbia University.

By Prof. Samuel A. Tucker.

For several years past practical work in electrochemistry has been conducted with a very modest installation placed in a portion of the Laboratory of Industrial Chemistry, but during the last year a very generous gift made possible the equipment of a laboratory devoted solely to the purpose of applied electrochemistry.

The present installation is situated in two rooms in the basement of Havemeyer Hall. Room 101 is 42 feet x 47 feet,

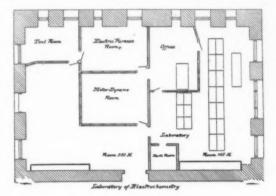


FIG. I .- PLAN OF LABORATORY.

and room 102 33 feet x 47 feet, thus a considerable space has been provided for the use of this laboratory.

The rooms have a ceiling 25 feet high, which gives good ventilation, and the students' working desks are so situated as to have an abundance of light from long windows on two sides of the room in which they are placed.

Room 101 has been divided up by a wood and glass partition into smaller rooms, and these consist of a motor-dynamo room, electric furnace room and tool room. At present a portion of the vacant space shown on the plan of room 101

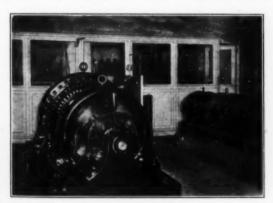


FIG. 2.-MOTOR-DYNAMO ROOM.

is occupied by some metallurgical furnaces, but it is expected that these will be removed, thus giving space for separate research rooms which are much desired.

In the motor-dynamo room are placed the generators for supplying the currents particularly adapted to the needs of the laboratory. These machines, together with the switchboard, were purchased from the Bullock Electric Co., of Cincinnati, and consists of two sets. The smaller comprise three direct-current dynamos directly connected to a 15-hp., 220-volt motor,

which takes current from the power house mains. The three dynamos of this set have separately excited fields, and are wound to give 4, 10 and 25 volts, with a capacity of 200 amps. each. The current is supplied to the laboratory by a four-wire system, giving these three voltages at the students' desks



FIG. 3.-50-KW. MOTOR-DYNAMO FOR ALTERNATING CURRENTS.

By making intermediate connections, voltages of 4, 10, 14, 25 35 and 39 are available. The switchboard for this machine is provided with the necessary switches, field rheostats and overload circuit breakers. The reading instruments are Weston volt and ammeter, arranged with dial switches, with shunts for the ammeter, so that measurements may be made on each of the three dynamos. This switchboard is also equipped with a double-throw switch, so that direct current from either the 10 or 25-volt machine may be connected with the special busbars in the electric furnace room.

The other machine is designed to supply heavy alternating currents for electric furnace work, and consists of a 75-hp., 220-volt direct-current motor, directly connected to a 50-kw.

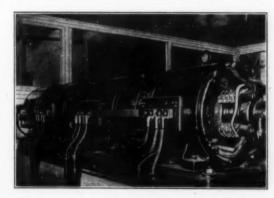


FIG. 4.—DIRECT-CURRENT MOTOR-DYNAMO SET.

alternator. The alternator is a single-phase, 60-cycle machine with revolving field, excited from the 110-volt circuit of the building. The armature wires come out to a panel on the machine, with appropriate switches for connecting them in series, multiple or parallel; and as the maximum voltage of the machine is 100, voltages of 25, 50, 75 and 100, with a current of 2,000 amps. at the low voltage, are thus available. The leads from the alternator connect directly to the switchboard in the electric furnace room. Arrangements are also made for supplying alternating current to the students' desks from this machine. The motor dynamos are set on concrete foundations, and there is practically no vibration. The smaller direct-current set was installed in preference to a storage battery, because the maintenance of the battery is a considerable

expense and requires very constant attention, although the first cost is somewhat in favor of the battery. Movable storage batteries are provided to meet the requirements of special cases.

The laboratory is provided with twenty double working-

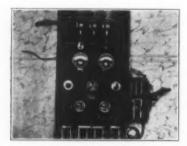


FIG. 5.—SWITCHBOARD FOR MOTOR-DYNAMO ROOM.

desks or tables for students, giving an unusually large working space. The tops are of Alberine stone, and each section is supplied with gas, water, blast and suction. An oak shelf supports the switchboards, which there is one for each student. These are of special design, enclosed in oak cases with glass

doors, thus keeping switch and other contacts away from chemical fumes. Two switches are mounted on each board, from which connection may readily be made to any of the six fused terminals. The currents provided are from the low-

voltage direct-current set as mentioned, and 110 volts direct current. These last terminals may be changed at will to alternating current, when the alternator is running, by means of a double-throw switch placed in the motor-dynamo room.

The working desks are lighted by half-shade lamps placed over each student's switchboard. The measuring instruments are the Weston portable. One ammeter and one voltmeter are supplied to each student at the beginning of his work. These give



FIG. 6.—SWITCHBOARD ON STUDENTS' DESKS.

three scale readings from zero to 3, 15 and 150 in 150 divisions of a volt, and in amperes from zero to 1.5, 15 and 150 in 150 divisions of an ampere by interchangeable shunts.



FIG. 7.-ELECTRIC FURNACE ROOM.

Suitable rheostats and electrolytic stands form part of the regular laboratory equipment. The electric wiring is carried through iron pipe in a very substantial manner, and the currents already named are carried to the hood at one end of the laboratory, as well as to a special stone table for the use of

larger apparatus than can be conveniently handled on the regular students' tables.

A set of platinum electrodes has been purchased for electrolytic work, as well as a number of small motors for stirring purposes. These motors are fitted with change gearing to give three speeds. The timepiece of the laboratory is a sixty-day regulator, with seconds beat-pendulum arranged

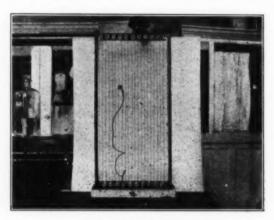


FIG. 8.—LOW-VOLTAGE RHEOSTAT.

with electrical contacts, so that the time factor in certain experiments may be accurately determined. A thermoelectric pyrometer by Siemens & Halske, reading to 1,500° C., and a Warner optical pyrometer for higher temperatures, have been included, as well as two chemical balances.

The electric furnace room adjoins the motor-dynamo room, and a new cement floor has been laid here, as the usual asphalt floors of the building are most unsuitable for this work. This room is enclosed by a galvanized roof, in order to keep the dust from the machinery in the adjoining rooms. On one side of the room is placed the switchboard controlling the bus-bars, which are 16 feet long, and of which there are three sets.

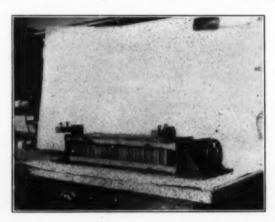


FIG. 9.—CARBON RHEOSTAT.

The bars are of aluminium, supplied especially for this purpose by the Pittsburg Reduction Co. The currents provided at these bus-bars are 200 amps., at 10 or 25 volts direct current from the small motor-dynamo set, 300 amps., at 110 volts direct current from the power house, and alternating current up to 2,000 amps. from the alternator. The switchboard is equipped with heavy switches, circuit breakers and measuring instruments. Connections are made from the bus-bars to the tables, on which the furnaces are set by means of one or more cables, depending upon the capacity of the furnace. The tables

are of wood, with fire-brick tops laid in cement. Separate switches, electrode holders, rheostats and measuring instruments complete the outfit for this part of the laboratory. The field rheostat for the alternator is operated from this room as well as from the motor-dynamo room, thus giving control of this machine while the furnaces are in operation.

The tool room is supplied with a 14-inch engine lathe, 16-inch shaper, sensitive drill and grinder, and a set of small tools used for repairs and for making appliances for the laboratory. Besides these tools, a heavy polishing and buffing head is installed in another part of the room, operated by an independent motor, and is used principally in electroplating work.

A brief description of some special appliances as used in the laboratory is mentioned, because these details are of importance to any one doing work in this subject.

For rheostats to be used in connection with low voltage and of a capacity of about 8 amps., the following was adopted: An iron frame work, 13 inches x 24 inches, mounted on a rectangular piece of Alberine stone, supports the porcelain insulators. The resistance wire is wound over these insulators, which are secured to the angle-iron cross pieces. The resistance wire is No. 15 german silver, and a sliding clip con-

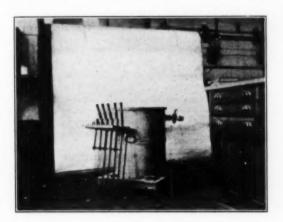


FIG. IO.-ARC FURNACE RHEOSTAT.

nection makes it possible to get fine adjustments of the current. Another form of rheostat designed for heavier currents is made up of thirty-five carbon plates, 4 inches x 7 inches x ½ inch, arranged in a frame so that pressure may be applied by the hand-wheel screw. The resistance is 3.3 ohms, and forms a convenient and compact apparatus for use up to 25 volts. Too high a voltage is apt to cause arcing with conse-

quent destruction of the plates.

For arc furnaces, a water-cooled resistance is in use, and has been a very practical form of rheostat. A slate top supports the bars and five switches, which are connected to copper bars, from which the resistance wire is strung. On each section there are three resistance wires, No. 17 1a, 1a, each 10½ feet long, spirial wound, each step being in parallel. The wires are water-cooled by placing in an ordinary galvanized ash can, means being provided for water circulation.

Each step has a resistance of about 1.5 ohms., and the carrying capacity, with all steps in, is about 400 amps.

A form of reciprocating stirrer for use in studying the deposits of copper under varying current density is shown. All connecting cables used in the electric furnace room are provided with brass tips and fit the various lugs and connections, which are all reamed to ½-inch holes.

The electrode holders are of iron for the most part, and will hold the electrode in a vertical or horizontal position at any desired elevation. The vertical adjustment is by a friction

clamp, which fits any of the uprights provided on the electric furnace tables.

The laboratory has been designed to provide facilities for the study of practical electrochemistry which is taken by chemists and electrical engineers, and also research work which is offered to graduates for higher degrees.

The courses offered in the laboratory are as follows:

1. Practical Electrochemistry.-Electroplating, influence of

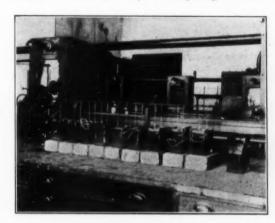


FIG. 11.-RECIPROCATING STIRRER FOR COPPER BATHS.

current density, temperature, concentration, etc., electrolysis with molten electrolytes, the use of diaphragms, study of typical industrial processes, electric furnace practice. This course comprises five afternoons of laboratory during one term, or its equivalent in two terms, and two lectures a week during one term on the applications of electrochemistry in the arts.

2. Practical Electrochemistry for Electrical Engineers.—Experiments in electric furnace work and the preparation of calcium carbide, aluminium, aluminium bronze, carborundum

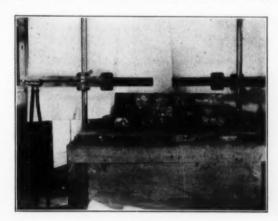


FIG. 12.—ELECTRODE HOLDERS.

and metallic chromium; illustrations of the wet processes in electrochemistry. This course is given one afternoon during part of the second term.

 Electrochemical Research.—Original investigation and research equivalent to 16 hours a week during one year.

4. Advanced Electrochemical Research.—Private study and research, original investigation and research during two years.

The following courses are in contemplation:

r. A practical course in the practice of electroplating and electrotyping, and the construction, maintenance and testing of storage batteries. 2. The preparation and study of the formation of typical

organic compounds by electrolysis.

Thus far research work has been done and investigations have been conducted in the laboratory on the following subjects: Potassium percarbonate, potassium chlorate, ammonium and potassium persulphate, electrolysis of calcium chloride with reference to the formation of chlorate, electric furnace for laboratory use, ethylene from inorganic sources, production of aluminium in the electric furnace with the aid of calcium carbide, some hitherto unknown metallic borides, production of abrasive compounds in the electric furnace, the study of polarization by alternating currents, the reduction of sodium nitrate by electrolysis, preparation of metallic lithium, formation of spongy lead for storage battery plates, are light electrodes and zirconium carbide, boron carbide, heat conserving qualities of certain refractory materials for electric furnace work, production of ferroalloys in the electric furnace, metallic calcium, kryptol and coke resistance furnaces, electrolysis of metallic oxides dissolved in boric oxide, electrolysis of chromium salts, measurement of temperature in resistance furnaces, melting points ot refractory materials used in electric furnace construction, resistance furnaces.

Factory Scale Experiments With Fused Electrolytes—II.

BY EDGAR A. ASHCROFT.

The principle of magnetic rotary agitation of the contents of the electrolytic cell, as described in my last article, has been successfully applied in two leading directions, and experiments have been more or less successfully conducted in others. The first of these applications is the direct electrolysis of sulphides (either mixed or singly). Several commercial processes are possible from this application of electrolysis accompanied by continuous mixing or agitation.

Fig. 3 represents the arrangement of plant for such a process, which consists in electrolyzing the ore suspended in any fused medium, such as halogen salt, between a solid indestructible anode and a fluid cathode, composed of the metal under recovery. Many forms and modifications of such a generally applicable process at once suggest themselves. In order to understand these more clearly the following reactions, all of which have been ascertained experimentally, should be studied:

Taking zinc chloride (most conveniently with a molecular proportion of sodium chloride, which for all practical purposes of reaction remains inert, although it substantially reduces the electrical resistance of the bath), we have results as follows:

Reaction A.—Zinc sulphide freely suspended does not react until a current is passed, when the net result of all reactions (probably very complicated in their intermediate forms), is that zinc sulphide is resolved into its elements, zinc and sulphur, which appear respectively at the cathode and the anode, thus:

ZnS + current = Zn (cathode) + S (anode).

If this operation is carried on at a temperature of about 450° C. the zinc is received molten and the sulphur distills as vapor, for the melting point of zinc and the vaporizing point of sulphur are nearly identical.

The original discovery of this reaction is due to J. Swinburne (British patent of 1898). This is a perfectly practical process for the treatment of pure blendes or those containing only silicious gangues (such as crystalline garnets, quartz, etc.), which are not affected by the zinc chloride electrolyte.

The theoretical e. m. f. of decomposition for zinc sulphide is only 0.9 volts, and the practical working e. m. f. will probably not exceed 2.5 volts. This method of treating blende ores has the great advantage that no preliminary roasting is necessary. When sulphuric acid is to be manufactured from the

blende simultaneously, it would be quite feasible to burn the sulphur, as it escapes from the cell by admitting air, and convey the SO₂ produced directly to the chambers (or other oxidizing device). In this way the expenses of fixing and operating either roasting plant or burners will be obviated, and the heat of combustion will be usefully employed in reducing the electrical energy necessary to maintain the heat of the fused charge. This process is very attractive at first sight, and has much to recommend it actually. But its limitations are greater than at first appear.

The desiderata for such a process of zinc recovery are:

- Ores free from any harmful impurity which would contaminate the electrolyte.
- Ores rich in blende and with little inert matter so as to avoid frequent cleaning out of the contents of the cell.
- Cheap electrical energy near to the ores and the markets.
 Efficient means for continuous agitation to prevent solid particles settling out on the cathode.

It is rather doubtful, in my opinion, in spite of the great apparent possibilities of such a process of blende reduction whether these conditions exist to-day or are at all likely to exist in future, except in a few small and isolated cases. The principal obstacles to commercial success are the impurities usually contained in the ore. For many ores contain both soluble gangue, such as horn blende and calcite, and also much iron manganese, etc., as sulphides, and few zinc ores which I have ever encountered are pure enough for practical success. Nevertheless, the study of these reactions is both interesting and may be important in other connections.

Reaction B.-Lead sulphide in a medium of fused zinc chloride reacts as follows:

 $ZnCl_2 + PS = ZnS + PbCl_3$

This reaction (the discovery of which, I believe, is originally due to Ganelin and Swinburne simultaneously) has been made the basis of several processes which we are not concerned with here.

On electrolyzing such a mixture, lead appears at the cathode (as long as any lead is present in the electrolyte), and, as before, sulphur distils at the anode. As lead is the base we are dealing with here, it is quite practicable to substitute lead chloride for the zinc chloride of the last experiment, and it is quite indifferent to the practical result whether this is done wholly or in part or not at all. Lead sulphide dissolves in lead chloride, forming a dark-colored fusible mass. Here we are getting much nearer to commercial conditions, for many rich deposits of galena exist, and in fact the ores of commerce suitable for the reverberatory furnace reduction methods are almost perfectly pure galena. Precaution must be taken, however, to minimize any reaction between the metallic cathode and the suspended sulphide (forming lower sulphides, etc.), and the best means to this end is to keep the total sulphur contents of both as low as possible. A great advantage exists for this method when silver has to be recovered in small quantities, for by working with two sets of cells (or periodically changing the cathode metal, which comes to the same thing) all the silver can be concentrated in a rich bullion, from which it can be cheaply recovered by direct cupellation, so that much of the expense of the ordinary refining process is eliminated.

I am of the opinion that in spite of the cheapness of the furnace method with which it has to compete, the electrolytic reduction of galena by this method is quite a commercial proposition, and may some day be extensively applied. Three considerations will tend to help in this direction:

- r. The growing scarcity of the purer galenas will bring more and more ore into the market, having some silver and some zinc contents (like the Broken Hill and Leadville lead concentrates), and the recovery of both zinc, silver, sulphur and lead are effected by the electrolytic method, whilst some of the silver and all of the zinc and sulphur in such concentrates is lost in the furnace process.
 - 2. The growing cheapness of electrical energy.

3. The device described in my last article for magnetically agitating the suspended particles of ore in the cell, which removes one of the great practical difficulties encountered by Swinburne and the writer when we first attempted to apply this idea as far back as 1899.

4. The progress of modern concentration methods which renders the emplete elimination of iron (the most trouble-some impurity) from the ore, in many cases now feasible. The theoretical e. m. f. for lead sulphide is only 0.4 volts, and the practical working e. m. f. when sulphur is also oxidized in the cell need not exceed 1.5 to 2.0 volts, so that, considering also the high ampere equivalent of lead, a very large output is possible from a given plant in comparison with other metals.

Reaction C.—Antimony sulphide (and many other sulphides, such as arsenic, etc., etc.,) reacts in a like manner with zinc chloride, thus:

$$Sb_2S_3 + 3ZnCl_2 = Sb_2Cl_2 + 3ZnS$$

Here a fortunate circumstance occurs, for the volatile chloride of antimony completely leaves the mixture (it vaporH.—The reactions of ferrous chloride with lead are especially interesting and important, as it has been found that they go opposite ways at different temperatures, and on this peculiarity the writer has based one method of eliminating iron in the mixed sulphide process. Thus, below 450° C.:

but at about 600° C. or less:

By making use of the latter reaction it is possible to remove all the iron from a mixture of fused chlorides by merely heating the mixture to a temperature of 600° C. with metallic lead, and agitating it well. The iron alloys with the excess of lead below, and as long as the alloy is slowly agitated and the contents of iron does not exceed some 5 per cent, it remains freely fluid. Afterwards the alloyed metal thus obtained may be treated by a settling out and skimming process, whereby a caked alloy of 25 per cent iron contents and a remainder of nearly pure lead is produced. The rich alloy of iron may be

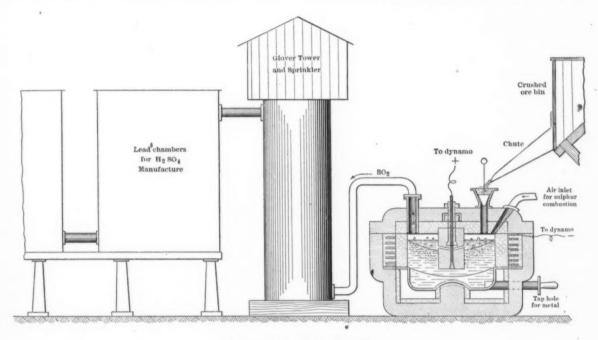


FIG. 3.-ARRANGEMENT OF PLANT.

izes at 200° C. and fuses at 70° C.), and may be separately condensed, and thus a ready means is obtained for eliminating a troublesome impurity which frequently occurs in lead ores. On this reaction have also been based several processes, having for their object the treatment of antimony and antimonial gold ores, but the consideration of the reactions of these would lead us too far afield from our subject.

Reaction D.—Copper (cuprous) and iron (ferrous) sulphides react with zinc chloride, but not completely or sharply. (This appears to be one of those cases where reactions go half way and then attain a state of equilibrium and stop.) As neither ferric nor cupric salts can exist in the presence of the sulphide reducing influence, the problem of the reaction of these metals is greatly simplified and falls under the following reactions E, F, G, H, etc.

A whole series of substitution reactions take place between metals and fused chlorides, the following of which are typical:

$$E$$
.—PbCl₂ + Zn = ZnCl₂ + Pb

$$F$$
. -2 AgCl + Pb = PbCl₃ + 2Ag

$$G$$
.— $FeCl_0 + Zn = ZnCl_0 + Fe$

melted in a cupola furnace and the lead recovered, and the nearly pure lead from which the rich alloy has been skimmed may be used again for precipitation of more iron. This process has importance when iron has to be removed from the fused bath as a preliminary to the precipitation of pure zinc, because iron can be much more cheaply removed from its alloy with lead than it can from its alloy with zinc. It is to be remembered, however, that for purposes of calculation in all such reactions an equivalent of electrical energy must be allowed corresponding to that required to precipitate the metal dissolved, or, in other words, corresponding to the atomicity of the iron removed.

Reaction J.—The more electropositive metals, of which sodium is the most important and typical, will practically precipitate all the others. Thus:

Reaction K.—Finally, sodium sulphide introduced into a bath of either zinc or lead chloride behaves in a similar way to lead sulphide when placed in zinc chloride; the chlorine

going to the more electropositive base and precipitating a sulphide of the lower metals. Thus:

 $Na_2S + PbCl_2 = 2NaCl + PbS$ $Na_2S + 7nCl = 2NaCl + 7nS$

 $Na_2S + ZnCl_2 = 2NaCl + ZnS$

The zinc and lead sulphides thus formed are of the lightcolored, finely divided variety, and are easily maintained in suspension and freely acted upon by the anodic products of nascent chlorine in the electrolytic process.

I have no doubt at all that the proper way to chemically regard all these reactions is to consider the direct electrolysis of zinc (or lead chloride) as the primary reaction, producing

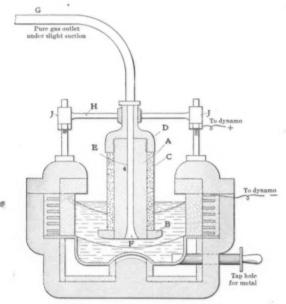


FIG. 4.-MODIFIED FORM OF ELECTROLYTIC CELL.

metal at the cathode and nascent chlorine at the anode which reacts with the suspended or dissolved particles of sulphide, setting free sulphur, reforming chlorides and simultaneously yielding up a quota of electrical energy. It was at one time supposed that iron salts were necessary as an intermediary for the exchange of chlorine to the sulphides, but this has been clearly proven not to be the case.

The second direction in which the principle of magnetic rotary agitation has found useful application is in connection with a new process of the writer's for cheap production of metallic sodium and some allied processes, making use of alloys with metallic lead and a double electrolytic apparatus with diverse electrolytes.

In my next article I will describe some applications of this double cell apparatus which was primarily devised for the sodium process, notice of which has already appeared in these columns.

In the meantime I will conclude this article with a description of a modification of the magnetic cell, which I have devised for use when (as sometimes happens) frequent and free access to the contents of the cell is necessary for economical working.

Fig. 4 illustrates such an apparatus, and the following references to the cut will explain its parts:

REFERENCES.

- A. Graphitized carbon block.
- B. Graphitized carbon shoe threaded to block.
- C. Protecting shield of fire-clay (tubular).
- D. Metal caps for carbon block.
- E. Porcelain-lined copper tube, making electrical connection with carbon and serving for gas passage.

- F. Spiral passages to assist projection of gas toward center of anode.
- G. Outlet for gas (maintained under slight suction).
- H. Yoke for support and adjustment of electrode.

J. Adjusting nuts.

By employing this form of modified anode and open electrolytic cell the gases (chlorine, etc.) can be caused to collect inwards in the center of the cell, and may be then withdrawn in a very rich and concentrated form, suitable for certain industrial processes where this condition is of great importance. The construction also affords free access to the cell for charging raw material and removing accumulations of dirt, etc., and allows of a ready adjustment of the position of the anode, both points of great importance in certain operations which I shall describe later.

Note.—By a slight oversight in Fig. 1 of my last article the lines showing the electrolyte in the zinc cell were omitted, and the words "tightened casting" in same figure should read "lightened casting."

Metallurgical Calculations.

By J. W. RICHARDS, PH. D.,

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CALCULATION OF THE CHARGE OF THE BLAST FURNACE.

In the last instalment we discussed the palance sheet of materials entering and leaving the furnace, showing the distribution of the ingredients of the charge and the blast into the various products and by-products of the furnace. We did not there go into the question as to how the proportions of the charge are determined by the metallurgist in charge of the furnace. There are, however, very few factors of the charge which can be controlled at will. The blast furnace reduces practically all of the iron present in the ore into the pig iron, and, therefore, if the ore contains 50 per cent of metallic iron and the pig iron 90 per cent, it will take 0.90 ÷ 0.50 = 180 parts of ore to furnish the iron in 100 parts of pig iron. The amount of ore to be used per unit of pig iron made is therefore fixed by the richness or poverty of the ore, and is not capable of variation. The amount of fuel used per unit of pig iron made is not fixed a priori, as is the amount of ore, but is governed by the calorific requirements of the furnace while in operation. If the pig iron and slag run colder than they should, it is evident that more heat must be put in or developed within the furnace, which the manager promptly proceeds to accomplish by increasing the temperature of the air blast (if he can), or by relatively increasing the amount of fuel in the charge (which he always can). The ratio of the weight of the ore and flux in the charge to the weight of fuel used is called the burden of the furnace, and in practice it is usual to charge at one time a fixed weight of fuel, and to vary the burden according to the heat requirements of the furnace. Changing the burden is therefore only another expression for changing the relative amount of fuel used, and this is varied simply from observation of the temperatures of iron and slag and the conclusions therefrom as to whether the burden is too heavy or unnecessarily light. The amount of blast used is another factor relatively fixed per unit of pig iron produced. Blow more blast, and more pig iron is produced; blow no blast (bank the furnace), and no pig iron is made; the ratio is not quite exact, but it is quite nearly true that, other conditions being equal, the output of pig iron is nearly proportional to the amount of blast blown. Variations in the temperatures of the blast produce important changes, which will be separately discussed.

The amount of flux used is really the one factor in which the manager has the greatest freedom of action. The amount of this indispensable substance used is determined by many factors, and can be varied between quite wide limits without fundamentally deranging the furnace. It is here a question of using sufficient flux in the charge to make with the unreduced constituents of ore and ash of the fuel a slag which shall be well-fused at the temperature of the furnace, shall carry off considerable sulphur, if much is present, and shall not corrode the lining of the furnace. These considerations are so important, and often so little understood, that we will discuss them more at length.

CALCULATION OF THE FLUX AND SLAG.

From the balance sheet of problem 51 it will be seen that the metallurgist running the Swedish blast furnace at Herräng, used, per 1,000 of pig iron made, 1530.2 of ore, 115.8 of limestone flux, 682 of charcoal, and blew in 2416.8 parts by weight of blast. It may with safety be believed that the amount of charcoal used was the minimum which he found by experience necessary to keep his furnace at proper temperature; and most American blast furnace managers will wonder how he could get along with so little. The amount of ore used was the necessary proportion to furnish the iron. The amount of blast was probably all that could be gotten out of the blowing apparatus with which the furnace was provided. Finally, the amount of flux was that amount necessary to make a proper slag. Let us investigate the question as to how its amount was determined and the characteristics of the proper slag.

The ingredients of the slag produced in the case in question are, from the balance sheet:

	From Ore.	From Flux.	From Fuel.	Total.
SiC ²	. 69.6	3.6	1.3	74.5
Al^2O^3	. 11.6	0.4	* * *	12.0
CaO	. 34.0	62.2	5.9	102.1
MgO	. 14.8	0.2	0.7	15.7
FeO	. 1.2	0.2	2.0	3.4
MnO	. 9.3	* * *		9.3
K2O			3.4	3.4
CaS	. 0,2	***	0.25	0.45
	140.7	66.6	13.55	220.85

And the percentage composition of the slag:

g

0

1.

36

15

ie

nf

SiO ²	=	33-73	Per Cent.	FeO	=	1.54	Per Cent.
Al ² O ³				MnO	=	4.21	66
CaO	=	46.23	66	K2O	=	1.54	"
MgO	=	7.11	44	CaS	_	0.20	66

The crucial question now presents itself, "What guided the metallurgist in choosing the quantity of limestone used, and in making a slag of the above composition?" The answer to this will develop the whole practice of fluxing.

Primarily, the fundamental guide in this matter is previous experience, as revealed in recorded analyses of slags which have worked properly. Such analyses have been made for a hundred years, and freely published; they show what compositions of slag have been found practicable and suitable in blast furnace practice. As reported by the chemists, the analyses show the varying percentages of SiO², Al²O³, CaO, MgO, FeO, MnO, etc., found in actual slags made in successful practice, and this information would be the metallurgist's guide in calculating the amount of flux to use to produce a good slag. On studying these analyses, we find that silica and lime are the predominating constituents of all blast furnace slags, the reason being that silica is the principal material to be fluxed, and that lime (from limestone) is the cheapest material which will flux it, and form a fusible slag.

Analyses of numerous blast furnace slags show the following variations of composition:

	0.4	eve	angre.	COLUMN CONTRACTOR CO.					
SiO ²	25	to	65	Per Cent.	FeO	0	to	6	Per Cent.
Al ² O ³	3	66	30	44	MnO	0	66	14	66
TiO ³	0	64	10	66	K10)				
CaO	12	66	50	64	Na2O	0	66	3	6.6
MgO			18		CaS	0	66	9	66

The above limits are not reached simultaneously by one and

the same slag. The ordinary variations may be summarized as follows (according to Ledebur):

	SiO ²	Al ² O ⁵	CaO = MgO
Producing gray iron, using			
charcoal	45 - 05	10-5	45 - 25
Producing gray iron, using			
coke	30 - 35	15 10	50 - 55
Producing white iron, using			
charcoal	45 - 50	10 - 5	45
Producing white iron, using			
coke	30 - 40	10 - 5	60 - 55
Producing spiegeleisen, using			
coke	30	16	55 - 45

In the latter case there is present 5—15 per cent of MnO. Among these varied compositions, however, there are various degrees of fusibility, and of suitability to the blast furnace's needs. The most easily fusible slags are those (considering only the main ingredients) which contain 35 per cent of lime, and in which, if alumina is present, each I per cent of alumina is balanced by the presence of 0.5 per cent additional lime. This rule gives good slags up to about 65 per cent of lime and alumina counted together. Another observation is, that with 33 to 40 per cent of lime in the slag, the amount of silica and

and in which, if alumina is present, each 1 per cent of alumina is balanced by the presence of 0.5 per cent additional lime. This rule gives good slags up to about 65 per cent of lime and alumina counted together. Another observation is, that with 33 to 40 per cent of lime in the slag, the amount of silica and alumina together being some 60 to 70 per cent, it is possible to change the relations of silica to alumina within large limits (i. e., from 40 to 50 silica and 20 alumina to 25 or 35 silica and 35 alumina) with very little effect upon the fusibility of the slag. On the other hand, with a low proportion of silica in the slag, say 30 to 40 per cent, it is equally possible to change the relations of lime to alumina within large limits (i. e., from 50 lime and 15 alumina to 35 lime and 35 alumina) with very little effect upon the fusibility of the slag.

The blast furnace manager usually decides upon the kind of slag he will make, on one of three or four assumptions:

(1) If there is little alumina present, and practically no magnesia, he usually assumes some ratio between the weights of silica and lime which he desires his slag to possess, and calculates the weight of flux necessary to make slag conforming to that condition. In charcoal furnaces, where there is practically no sulphur in the charge, the silica may be 1.5 to 2.0 times the lime present; in coke furnaces, where it is necessary to eliminate much sulphur, this ratio is usually 0.5 to 1.0.

Illustration: If a ton of iron ore carries 300 pounds of silica, how much limestone, which is practically pure calcium carbonate, will be required to slag it, neglecting flux necessary to slag the ash of the fuel?

Solution: Pure limestone is $CaCO^3$, or $CaO \cdot CO^3$, and carries 56 per cent of lime, which will go into the slag, and 44 per cent of carbonic oxide (CO^3), which goes into the gases. Using x pounds of limestone, the weight of lime going into the slag is 0.56 x. If we assume the ratio of silica to lime = 1 for a coke furnace, and 1.75 for a charcoal furnace, we get the two equations and corresponding values of x:

$$\frac{300}{0.56} = 1$$
 whence $x = 536$ Pounds.
 $\frac{300}{0.56} = 1.75$ whence $x = 307$ "

(2) If considerable magnesia is present it is usual to either count it simply as so much lime, or else to calculate the weight of lime to which it is chemically equivalent, and add this to the lime, calling the sum the "summated lime." The ratio of silica to lime is then used as the ratio of silica to lime plus magnesia, or of silica to summated lime. When there is considerable magnesia present the chemical summation should always be made. Small amounts of MnO, FeO and K²O or Na²O are also chemically summated as lime. The chemical summation is based on the fact that one molecule of a base containing one atom of oxygen is considered the equivalent in fluxing power

of any other similar molecule; e. g., CaO, MgO, FeO, MnO, K2O, Na3O are considered as equivalent; but as these molecules weigh differently, we have equivalent fluxing powers in the following weights:

																15											
CaO .				,	,	,			,	*		,		÷		,			,							,	50
MgO				,			,	,		,				,	,		×	,	5			*		ě		5	40
FeO .		+		×		,	,	,			*		*	,	,	4	r	*		ě	,	,			×	×	72
MnO																											71
K2O .	(8)	×	,					×		×		×		×	×		,		×	×			×	8		*	94
Na®O													0				0	0								0	62

from which we conclude that since 40 parts by weight of magnesia is equivalent to 56 parts of lime, that, therefore, the

lime equivalent of any weight of magnesia is $\frac{56}{40}$ of the

weight of the magnesia. Similarly, we get the lime equivalent of these bases as follows:

CaO equivalent of any weight of MgO =
$$\frac{56}{-} \times$$
 weight MgO = $\frac{56}{-} \times$ weight MgO = $\frac{56}{-} \times$ " FeO = $\frac{56}{-} \times$ " FeO = $\frac{56}{-} \times$ " MnO = $\frac{56}{-} \times$ " MnO = $\frac{56}{-} \times$ " K³O = $\frac{56}{-} \times$ " K³O = $\frac{56}{-} \times$ " Na³O = $\frac{56}{-62} \times$ " Na³O

Illustration: An iron ore contributes to the slag 350 pounds of silica, 12 of FeO and 60 of MnO. It is desired to flux it by using limestone containing 38.1 per cent lime, 13.6 magnesia, 3.4 silica, and 44.9 carbonic oxide (CO*). How much flux must be used to produce a ratio of silica to summated lime = 0.8?

Solution: Letting x be the pounds of limestone used, the ingredients of the slag will be

$$SiO^3$$
 = 350 + 0.034 x
 CaO = 0.381 x
 MgO = 0.136 x
 FeO = 12
 MnO = 60

The lime equivalents of the MgO, FeO and MnO are:

CaO equivalent of MgO = 0.136
$$x \times \frac{56}{40}$$
 = 0.1904 x

"FeO = 12 $\times \frac{56}{72}$ = 9.33

"MnO = 60 $\times \frac{56}{71}$ = 47.32
"CaO = 0.381 $x \times 1$ = 0.381 x

Summated CaO =
$$0.5714 x + 56.65$$

The ratio of silica to summated lime is therefore:

$$\frac{350 + 0.034 \ x}{56.65 + 0.5714 \ x} = 0.8$$
Whence $x = 720$ pounds.

It is easily seen that this method of solution, calling x the weight of flux used, and then getting expressions for the weight of each ingredient in the slag and the weight of the whole slag, is a very general solution which is applicable to any

kind of assumed composition to which it is desired that the slag shall conform.

(3) If alumina is present in the slag-forming constituents of the ore it also may be reckoned with in several ways. It may be reckoned as so much by weight, and added in as such to the silica or the lime, or it may be calculated to its silica or its lime equivalent, and added into the summated silica or the summated lime. Here we touch on a question which has agitated blast furnace managers and theorists for a generation: Should the alumina be reckoned with the bases or with the acids; summated as silica or as lime? It would be presumptuous to set forth a dictum on a subject which has been so long and so ably discussed by some of the best iron metallurgists, but we will assume, as somewhere near the truth, that as far as the elmination of sulphur in the slag is concerned, alumina acts in slags low in silica as though it were lime, not in the proportions of its lime equivalent

$$\left(=\frac{168}{102} \times \text{weight of alumina}\right)$$

but rather in about the proportions of its simple weight. As far as fusibility is concerned, in high silica slags alumina increases the fusibility up to a certain point, above which it decreases it. It acts in these, therefore, like lime, and may be classed with the bases. In low silica slags, below 45 per cent, alumina acts like silica when considerable is present, and like lime when less is present; for instance, in a low lime, high alumina slag, alumina and silica may be substituted one for the other within wide limits, without materially affecting the fusibility of the slag; in a high lime, high alumina slag, alumina and lime may be substituted for each other within wide limits without sensibly changing the fusibility. To state the matter as succintly as possible, in slags low in silica (30 to 35 per cent), alumina reinforces the bases in the elimination of sulphur; in regard to fusibility, it acts like silica in a slag low in both silica and lime, and like lime in all other blast furnace slags.

Illustration: An iron ore carries 10 per cent of its weight of silica and 6 per cent of alumina. The limestone on hand contains 37.3 per cent lime, 13.3 magnesia, 3.3 silica, 44 carbonic oxide (CO^2), and 2.1 per cent alumina. How much flux is required per 1,000 parts of ore to make (a) a slag with 49 per cent of silica plus alumina; (b) a slag with 33 per cent silica; (c) a slag with summated silica = the summated lime?

Solution:

Returning to the slag resulting from the proportions of flux chosen in Problem 51, and containing:

SiO ³	33-74	Per Cent.	FeO	1.54	Per Cent.
Al ² O ³	5-43	64	MnO	4.21	43.
CaO	46.23	÷1	· K ² O	1.54	**
MgO	7.11	44	CaS	0.20	**

We see that its percentage of silica is low, therefore it is adapted to produce pig iron low in sulphur; its percentage of alumina is low, and therefore its presence increases the fusibility of the slag, which would otherwise be rather deficient, because of the high lime and somewhat considerable amount of other bases. In such a slag, alumina would be summated with the bases.

with the bases.	
The natio of cilies to become	33.74 = 0.516
The ratio of silica to bases is	66.06
	33.74
The ratio of silica to lime plus magnesia is	= 0.632
	53-33
The summated lime	=46.23
168	
+ CaO equivalent of Al ² O ³ = (5.43)	= 8.94
102	
56	
+ CaO equivalent of MgO = $$ (7.11)	= 9.95
40	
56	
+ CaO equivalent of FeO = - (1.54)	= 1.20
72	
+ CaO equivalent of MnO = $\frac{56}{-}$ (4.21)	- 222
+ CaO equivalent of MilO = (4.21)	= 3.32
+ CaO equivalent of $K^{2}O = \frac{56}{}$ (1.54)	- 002
94	
7	***
	70.56
33-74	
Ratio silica to summated lime =	= 0.478
70.56	

Problem 52.

In a blast furnace charge, consisting of 1530.2 pounds of ore and 682 pounds of charcoal, per 1,000 pounds of pig iron made, it is known from the balance sheet of above materials that they will contribute to the slag the following slag-forming ingredients. (See balance sheet, Problem 51):

SiO ³	70.9 Po	unds.	FeO	3.2	Pounds.
Al ² O ³	11.6	44	MnO	9.3	64
CaO	39.9	44	K ² O	3-4	44
MgO	15.5	44	CaS	0.4	66

The limestone at hand contains:

CaO	53-74	Per Cent.	Al ² O ³	0.32	Per Cent.
MgO	0.17	48	Fe2O2	0.18	44
SiO ²	3.14	64	CO2	42.42	44

Required: The weight of limestone to be used to make: (1) A slag containing 33.74 per cent of silica. (2) A slag in which the ratio of silica to bases is 0.516. (3) A slag in which the ratio silica to summated lime is 0.478.

Solution: This problem embodies the conditions which confront the metallurgist when desiring to calculate the flux needed by any given furnace, and we have assumed certain working ratios to be aimed at in the slag, in order to elucidate the method of solution.

CONSTITUENTS OF SLAG.

From Ore		
and Fuel.	From Flux.	Total.
SiO ² 70.9	0.0314 x	70.9 + 0.0314 x
Al ² O ³ 11.6	0.0032 x	11.6 + 0.0032 x
CaO 30.0	0.5374 x	$30.0 \pm 0.5374 \text{ r}$

MgO	15.5	0.0017 x	15.5 + 0.0017 x
FeO	3.2	144 (0.0018 x)	3.2 + 0.0016 x
MnO	9.3		9.3
K2O	3-4		3.4
CaS	0.4		0.4
	154.2	0.5753 x	154.2 + 0.5753 x

(1) To make a slag with 33.74 per cent of silica we must have

$$70.9 + 0.0314 x = 0.3374 (154.2 + 0.5753 x)$$

whence $x = 116$ pounds.

(2) To make a slag with ratio of silica to bases 0.516 we must have

$$70.9 + 0.0314 x = 0.516 (82.9 + 0.5439 x)$$

whence $x = 116$ pounds.

(3) To make a slag with ratio of silica to summated lime

0.478, we must first summate the lime as follows:

Lime
$$= 39.9 + 0.5374 x$$
Lime equiv. of Al²O² = $\frac{168}{-102}(11.6 + 0.0032 x) = 19.1 + 0.0053 x$

" MgO = $\frac{56}{-40}(15.5 + 0.0017 x) = 21.7 + 0.0024 x$

" FeO = $\frac{56}{72}(3.2 + 0.0016 x) = 2.5 + 0.0012 x$

" MnO = $\frac{56}{71}(9.3) = 7.3$

" K²O = $\frac{56}{94}(3.4) = 2.0$

Summated lime = 92.5 + 0.5463 x

therefore 70.9 + 0.0314 x = 0.478 (92.5 + 0.5463 x) whence x = 116 pounds.

COMPARISON OF FUELS, FLUXES AND ORES.

By properly utilizing the preceding principles, it is possible to compare different varieties of fuels, fluxes or ores with each other, and thus to determine their relative values to the furnace, as far as can be inferred from their chemical composition. (Some of the following methods are from a paper by Mr. F. W. Gordon; Trans. Am. Institute Mining Eng., 1892, p. 61.)

COMPARISON OF FUELS.

If different qualities of fuel are available it is possible to calculate which is the most advantageous to use in the furnace. The fixed carbon only is efficient for the furnace, and not all of that, because the ash of the fuel needs to be fluxed to slag, and a certain amount of the fixed carbon will need to be burned simply to melt this slag. Then the cost of the limestone to be used to flux this ash must be counted in, and finally part of the labor costs of running the furnace must be charged against the slag. We can thus calculate the total charges against the fuel to supply one part of available carbon, which is the best basis upon which to compare different fuels.

Illustration: Two varieties of coke are available for a blast furnace, analyzing respectively:

		No. 1.		No. 2.
Fixed carbon	84	Per Cent.	90	Per Cent.
Volatile matter	.2	44	1	44
Moisture	5	44	3	**
Ash	0	6.5	6	**

And costing respectively \$4.50 and \$5.50 per ton. The ash of the fuels analyzes respectively:

	No. I.	No. 2.
Silica	55 Per Cent.	25 Per Cent.
Alumina	25 "	5 "
Lime	15 "	50 "
Magnesia	5 "	10 "
Ferric oxide		10 "

They are to be fluxed with the limestone of preceding problem, assumed to cost \$1.00 per ton, and to make a slag carrying 40 per cent of silica and alumina together. Assume an average of 0.228 parts of fixed carbon necessary to melt down I part of slag, and that the manufacturing costs borne by the slag amount to \$1.00 per ton. What are the relative values of the two fuels to this furnace?

Solution: The amounts of flux needed to 100 parts of each fuel burned will be found as follows, letting x be the amount of flux used:

	Slag No. 1.	Slag No. 2.
Silica	4.95 + 0.0314x	1.50 + 0.0314 x
Alumina	2.25 + 0.0032 x	0.30 + 0.0032 x
Lime	1.35 + 0.5374x	3.00 + 0.5374 x
Magnesia	0.45 + 0.0017 x	0.60 + 0.0017 x
Ferrous oxide.	0.0016 x	0.54 + 0.0016 x

Total weights. 9.00 + 0.5753 x 5.94 + 0.5753 x

Therefore, in case No. 1:

$$\begin{array}{c} 7.20 + 0.0346 \, x = 0.40 \, (9.00 + 0.5753 \, x) \\ x = 18.4 \end{array}$$
 whence

And in case No. 2:

No. I.

$$1.80 + 0.0346 x = 0.40 (9.00 + 0.5753 x)$$

nence $x = -9.2$

The negative value in case No. 2 simply means that the ash of fuel No. 2 is more basic than the slag and, therefore, requires no limestone, but itself acts as a basic flux. Per ton of fuel burned, there would be required respectively 0.184 and - 0.092 tons of limestone, and the weights of slag would be 0.196 tons and 0.0065 tons. (Substituting values of x in the total weights of slags.)

The weights of fixed carbon necessary to smelt these weights of slag would be:

No. 1.
$$0.196 \times 0.228 = 0.0447$$
 Tons.
No. 2. $0.0065 \times 0.228 = 0.0015$ "

Leaving as available fixed carbon for the furnace in each case:

No. 1.
$$0.84 - 0.0447 = 0.7953$$
 Tons.
No. 2. $0.90 - 0.0015 = 0.8985$ "

From these figures the cost of 1 ton of available carbon furnished by each fuel, adding in manufacturing cost chargeable against the slag, is

Cost of coke, \$4.50 ÷ 0.7953	= \$5.658
Cost of limestone, $$1.00 \times 0.184 \div 0.7953$	= 0.231
Costs against slag, $$1.00 \times 0.196 \div 0.7953$	= 0.246
No. 2	\$6.135
Cost of coke, \$5.50 ÷ 0.8085	= \$6.233
	1 00
Cost of limestone, $$1.00 \times (-0.092) \div 0.89$	85 = -0.102
Cost against slag, $1.00 \times 0.0065 \times 0.8985$	= 0.007

The two fuels, at the prices given, are therefore of almost exactly the same value to the furnace.

\$6.138

The solution along the lines shown is general, for any desired composition of slag, or for use with any given limestone, and is a valuable means of comparing the values of different fuels. The cost of a ton of pure, available carbon, when furnished by any given fuel, is an item which is useful when com-

paring the relative values of different fluxes or ores with each other.

COMPARISON OF FLUXES.

If different qualities of flux are available it is very desirable to be able to calculate which is the most economical to use in the furnace. Any acid ingredients in the flux diminish very sharply its efficient fluxing power, because they must first be satisfied from the bases present in the same proportions as acid to bases in the final slag. The slag thus formed from the impurities requires further to be melted, and other costs are properly chargeable against it. The best comparison is finally obtained by calculating for each flux available the cost from it of pure net lime, or net summated lime, analogous to the calculation for pure net carbon in the case of fuels. Any ordinary condition may be imposed upon the slag which the furnace is to produce.

Illustration: There are available for a furnace two qualities of limestone, containing respectively:

CaO	53.74 Pc	er Cent.	47.80 P	er Cent.
MgO	0.17	86	4.61	66
SiO ²	3.14	6.6	5.12	**
Al ² O ²	0.32	66	3.36	66
Fe2O3	0.18	66	1.10	6.6
CO2	42.42	46	37-55	66

The first costs \$1.00 per ton, the second \$0.80. Assume them smelted with fuel furnishing pure available fixed carbon at \$6.135 per ton; that 0.228 tons of pure fixed carbon is needed to smelt 1 ton of slag; that manufacturing costs against slag are \$1.00 per ton, and that the slag to be made in the furnace must have summated silica equal to summated lime. Compare the relative values of the two fluxes.

Solution: We will direct our calculations towards finding the net cost of I ton of pure available summated lime from each of the two limestones. The summated lime and silica in each flux are:

	No. I.	NO. 2.
Summated lime	0.5411	0.5502
Summated silica	0.0342	0.0808
Excess of summated lime	0.5069	0.4694

The weights of slag formed from the impurities present in each limestone will be:

	140. 1.	140. 2.
CaO (difference between amount present		
and excess of summated lime found)	0.0305	0.0086
MgO	0.0017	0.0461
FeO	0.0016	0.0099
SiO ²	0.0314	0.0512
Al ² O ³	0.0032	0.0336
21		
Totals	0.0684	0.1494

The cost of I ton of pure available lime from each of these fluxes will therefore be, adding in costs chargeable against the slags formed by the impurities present:

No. 1:		
Cost of limestone, \$1.00 ÷ 0.5069	=	\$1.973
Cost of carbon for melting slag, $6.135 \times 0.228 \times 0.0684 \div 0.5069$	=	0.188
Costs of running, chargeable against slag, \$1.00 X		
$0.0684 \div 0.5069$	=	0.135
No. 2:		\$2.296
Cost of limestone, \$0.80 ÷ 0.4694	=	1.904
Cost of carbon for melting slag, \$6.135 × 0.228 ×		
0.1494 ÷ 0.4694	=	0.465
Cost of running, chargeable against slag, \$1.00 X		
0.1494 ÷ 0.4694	-	0.318
		\$2 487

The conclusion is that the poorer limestone, at \$0.20 per ton less cost, is in reality costing \$0.191 per ton more for pure available lime, or is in reality 8.3 per cent dearer than the first, instead of being 20 per cent cheaper.

The method of calculation here described is quite general for any compositions of limestone or other flux, and for any assumed conditions which the slag must conform to.

COMPARISON OF ORES.

As the more complicated, we come to the comparison of various ores which may be at the iron master's disposal. Here a similar method of procedure is advisable. It can be calculated first, for a unit weight of ore, how much pure lime would be required to flux its impurities, how much pure carbon would be required to melt the slag thus formed, and to the costs of each of these would be added the handling of the slag. Each of these can be also expressed per unit of pure oxide of iron in the ore; and if to their sum we add the cost of ore necessary to furnish unit weight of pure oxide of iron we obtain the total costs per unit weight of pure iron oxide (Fe2O3). This is the basis on which different ores may then be compared. It must not be forgotten that one ore may, because of higher sulphur content, require the production of a more basic slag, so that the amounts of flux required and slag formed will be influenced by the condition necessary to impose on the slag in each case.

Illustration: The iron ore briquettes (of Problem 51) contained Fe²O³, 85.93 per cent; FeO, 3.96; SiO³, 5.50; MnO, 0.63; Al²O³, 0.76; CaQ, 2.23; MgO, 0.97 per cent. If these cost \$4.40 per ton, and are smelted in a furnace making slag with ratio of silica to bases 0.516, and assuming 0.3 per cent of the iron, 82.7 per cent of the silica, and 96.6 per cent of the manganese to go into the slag, what is the cost per ton of pure Fe³O³ from this source, charging pure lime for fluxing at \$2.296 per ton, pure carbon for smelting slag at \$6.135 per ton, and requiring 0.228 tons of carbon for one of slag, adding also manufacturing costs at \$1.00 per ton of slag?

Solution: The slag-forming ingredients from 1 ton of ore briquettes are:

FeO
$$0.003 \times \frac{72}{56} \times \frac{112}{56}$$

$$\left(0.8593 \times \frac{112}{160}\right) + \left(0.0396 \times \frac{56}{72}\right) = 0.0008 \text{ Tons.}$$

MnO $0.966 \times 0.0063 = 0.0023 \text{ "}$

MgO 0.0097 "

Al^{70 *}

and the bases to satisfy the silica present must be $0.0455 \div 0.516 = 0.0455 \text{ "}$

and the bases already present ended and total weight of slag 0.0455 "

Efficient Fe²O³ in 1 ton of ore plus Fe³O³ equivalent of reduced FeO = $0.0396 \times 0.0432 \text{ "}$

Total available Fe³O³ = 0.0025 "

Cost of 1 ton pure available Fe2O3 from these briquettes:

Costs chargeable against slag, \$1.00 \times 0.1337 \div 0.9025 = 0.148

= \$4.875

= 0.207

Total = \$5.336

0.106

Cost of ore, \$4.40 ÷ 0.9025

 $0.1337 \div 0.9025$

Cost of pure lime, \$2.296 × 0.0417 ÷ 0.9025

Cost of carbon for melting slag, $\$6.135 \times 0.228 \times$

A similar calculation is possible with any ore of any given composition, and making any assumed quality of slag. The costs of 1 ton of pure ferric oxide thus calculated will give the relative costs of the iron obtained from these different sources, and therefore indicate the relative values of the different ores to the blast furnace manager.

Problem 53.

Assume a blast furnace manager to use the ore of preceding illustration, furnishing pure Fe⁸O² at a net cost of \$5.336 per ton, and to use with it fuel furnishing pure carbon at \$6.135 per ton, there being required for reduction and melting the iron produced and furnishing it with carbon, 0.66 tons of pure available carbon per ton of pig iron produced, and the pig iron containing 96.656 per cent of iron. The running costs of the furnace are \$3.00 per ton of pig iron produced (costs against slag not included).

Required: The cost of the pig iron per ton.

Solution:	и.		
Fe 2 O 8 required 160 \div 112 \times 0.96656	=	1.3808	Tons
Cost of the ore, $\$5.336 \times 1.3808$	=	\$7.368	
Cost of fuel, \$6.135 × 0.66	=	4.049	
Cost of manufacturing (share against pig iron	n)=	2.000	
Total co		Q	
1 Otal Cos	31	Φ13-417	

Storage Batteries.

A paper, recently presented by Dr. Rudolf Gahl before the Colorado Scientific Society, deals with the scientific principles of electric storage batteries and contains much that is interesting. The author discusses many points in a novel way, basing his discussion throughout on the principles of modern physical chemistry. Dr. Gahl is now a consulting chemist and engineer in Denver, Col., having been formerly connected with the two most prominent storage battery factories in this country and in Germany. He has carried out a large amount of research work in connection with storage batteries.

The first part of the paper is a general discussion of the possibilities of storing electrical energy in form of chemical energy. Dr. Gahl derives several axioms from practical experience. For instance, that a practical accumulator cannot be based on gas reactions at present; that an electric storage battery must contain only one liquid, and then, of course, no diaphragm; that soluble cathodes for storage cells are not reliable (which excludes the zinc-lead cell).

The discussion thus narrows down to cells containing insoluble cathodes and insoluble anodes. The electrolyte must not be decomposed by the current into gaseous products; it must have a high conductivity; it must be cheap. Among the acid electrolytes sulphuric acid stands alone. Since only barium and lead form insoluble salts with sulphuric acid and since barium has to be excluded, being non-reducible by the electric current, the lead cell is the only practical acid cell. For alkaline cells potassium and sodium hydrate are suitable electrolytes, while iron and cadmium may be used for cathodes and the oxides of nickel, iron, silver for the anodes. For the support of the plates lead must be used in acid solutions and nickel in alkaline solutions.

Dr. Gahl then gave a concise review of the present situation of the lead storage battery. He described the methods of Plante and of Brush-Faure.

"Plante plates are used either as positive plates or as negative plates. One advantage of using these Plante plates for the negative, that is, for the spongy lead plates, is that after a while, owing to reactions which have not been studied thoroughly enough, the spongy lead loses its consistency and solidifies into solid lead. Of course, there is a decrease in the capacity of these plates according to the change in their

structure. A remedy is available in such a case, which is to use the plates which have been negative before, as positive plates, the effect of a long series of discharges and charges on the positive Plante plates being just the opposite from its

effect on the negative plates.

"The solid body of the positive plate is slowly but constantly attacked by the combined effect of sulphuric acid and current, and the capacity would increase considerably unless through the action of the gas bubbles developed on the charge of the battery some lead peroxide would be loosened and fall off. But, nevertheless, they are not apt to lose their capacity as quick as the negative plates, and on reversing the polarity and using the negative plates as positives and the positive ones as negative ones, the newly-made negative plates start in with quite a high capacity and the positive plates regain their former capacity through the attack to which positive plates are always exposed.

"The grids in American Plante plates are cut mechanically The leading Plante plates in Europe are cast by machines. plates, which do not seem to gain favor in this country, possibly because casting of such fine things as these battery plates requires a high development of the art of casting, to which art the younger American factories have not paid enough at-

"Positive Faure-Brush plates are not made by any prominent plant in this country, except in small sizes for automobile work; in fact, it is very difficult to make them, but it is possible, for some good plates of this type are manufactured abroad. Negative Faure plates are made in this country by the National Battery Co., Buffalo, N. Y., and by the Electric Storage Battery Co., Philadelphia, Pa. The negative Brush plates formerly had the same disadvantage which the Plante plates still have, of losing their capacity after some time of service through shrinkage, but it has been found that by giving the paste used for manufacturing these plates a certain composition, that tendency for shrinkage can be overcome, and that it is even possible to manufacture plates which expand and become more and more porous after they have worked some time. The expansion, and accordingly the porosity of the plates, had to be kept down to prevent falling off of the active material.

Recently another type has been brought on the market by the Tudor companies in Europe and by the Philadelphia Electric Storage Battery Co. in this country, which permits the use of material of high expanding power, the falling off of active mass being prevented by a kind of perforated lead wrapper, cast together with the grid. This plate is called box plate, and means a vast improvement over the negative plates made before, and it evidently seems to solve the problem of negative plates, for now the negative plates have a longer life than the positive plates, whereas, formerly the life of the

battery was always limited by the negative plates."

For the choice of the best concentration of the acid several considerations are of importance. The higher the concentration, the higher the e. m. f. and the greater the amount of available e. m. f. per unit of weight. On the other hand, the conductivity of sulphuric acid has a maximum near 1.2 specific gravity, which is about the standard in practical use. Most important, however, is the regard for the life of the plate, since a variation in one direction is harmful to a negative plate and a variation in the other direction to a

The capacity of the storage plates is limited, not by the absolute amount of the acid in the cell, but the quantity of acid in the immediate neighborhood of the working plate. It is most important to make the plates as porous as possible, so as to accelerate diffusion of fresh acid into the batteries where

the acid becomes exhausted during discharge.

"The construction of the box plate, which I mentioned before, is practically the result of investigations in this line; it is a pity that the qualities of lead peroxide do not permit these

plates to be made as porous as would be desirable for increasing their capacity. As lead peroxide has a tendency to disintegrate in fine particles, it is not possible to make a porous positive plate without endangering its life very much. This is the reason why the construction of an efficient and at the same time durable positive plate presents so many difficulties; in fact, is the main difficulty at present in raising the life of the lead storage cell to the life of other electrical machines, as, for example, a dynamo or motor."

After a short explanation of the delecterious effect of the presence of iron and platinum in the electrolyte on the capacity of the cell, Dr. Gahl discussed very briefly the possibilities of the alkaline cell, in which field "Edison, the great American inventor, seems to have been at least partly successful."

Both cadmium and silver make very good plates for high capacity, although the e. m. f. of the combination is only about I volt. For the combination of iron with nickel the voltage is somewhat higher, but probably owing to the small conductivity of the oxides, it is not possible to construct plates of any thickness, even when the conductivity of the plates is increased by the addition of some good conductor like graphite or some metallic substance. The problem of making suitable iron and nickel electrodes is, therefore, chiefly a mechanical

"But the problem of a light alkaline cell will enter a new phase, when it will be commercially possible to use cadmium. It does not seem to be doubtful to me that as soon as large enough quantities of cadmium are discovered the problem of a light accumulator for automobiles will be brought very near its solution, at least for cases where the price does not cut so very large a figure as to exclude the application of metals like cadmium and silver.

"Perhaps it is left to Colorado to make this development possible by supplying sufficient amounts of cadmium."

Notes on Electrochemistry and Metallurgy in Great Britain

(From Our Special Correspondent.)

THE HERMITE ELECTROLYTIC HYPOCHLORITE PLANT AT POPLAR.

The first account which has appeared in an English journal of this installation, which has been installed by the Public Health Authority, is that in the Lancet, the celebrated medical journal. It is, therefore, not surprising that electrochemical details are not so full as might be desired. For instance, the specific gravity of the solution and the respective proportions of sodium chloride and magnesium chloride are not stated, neither is the current density mentioned. The electrolyzer, which is arranged on a cascade principle, apparently consists of forty-eight cells, arranged in four tiers of twelve cells each. The solution passes through glass tubing from cell to cell, and from stage to stage, the strength increasing until, at the outflow, the solution contains about 5 grammes of available chlorine per liter. The output is 185 gallons per day, the voltage at the tank terminals is 240 volts, and the current 15 amps., from which data it may be calculated that the yield of available chlorine is 72 grammes per kw-hour. Now, by no stretch of imagination (even of an inventor's elastic kind) can this be considered favorable on the side of electrochemical efficiency. In the March issue of ELECTROCHEMICAL AND METALLURGICAL INDUSTRY are chronicled outputs at this chlorine strength exceeding 200 grammes of available chlorine per kw-hour.

On the score of stability the resultant output is far superior to that chronicled by Sir Henry Roscoe in his investigation of the old Worthing installation. Some tests in the Lancet laboratory showed a depreciation of 5 per cent in five weeks for a solution containing 5.68 grammes of available chlorine per liter, and slightly higher losses for weaker solutions.

THE INSTITUTION OF MINING AND METALLURGY.
CYANIDE PROCESS.

The papers read before this institution at its last meeting, March 15, included a "Note on the Ammonia-Copper Cyanide Process," by Mr. E. Le G. Brereton; a paper on the "Cyanide Treatment of Cupriferous Tailings by the Sulphuric Acid Process," by Mr. W. S. Brown. There was also a paper on "Earth Temperature on the Witwaters Rand Gold Fields and Their Relation to Deep Level Mining," by Mr. H. F. Marriott.

Mr. Brereton's paper is a recitation of some experiments, from which he concludes that "under certain conditions cuprous potassium cyanide in an ammoniacal solution is broken up with the liberation of an alkaline cyanide." Omitting the details of the actual experiments the following are the author's detailed conclusions:

"I. That there is always free cyanide (as indicated by AgNO₃) in the solution when the blue solutions are decolorized. II. The free cyanide indicated by AgNO₃ increases in the solution on standing. III. The removal of free cyanide by AgNO₃ tends to cause the solutions to regain the blue color, the precipitate of silver iodide dissolving, and a free cyanide again making its appearance. IV. That a large excess of ammonium hydrate in the solution is necessary if this reaction is to be carried to its limits. When an ammoniacal copper solution is titrated with a potassium cyanide solution, free cyanide is indicated by silver nitrate at the earliest stages of the titration.

"In the above experiments the change of the solution from colorless to blue, and the quantity of silver which has passed into solution, presumably as a double cyanide, is clear evidence of chemical changes taking place in the solution."

Mr. Brereton is of the opinion that in a strongly ammoniacal solution containing also an ammonium salt Cu₂Cy₂xKCy (assuming that such a compound is formed, which seems probable), is unstable unless an excess of free alkaline cyanide is present. If this excess of free alkaline cyanide is removed a reaction takes place of possibly somewhat the following order:

$$\begin{array}{l} 2(\text{Cu}_2\text{Cy}_2.2\text{KCy}) + \text{12NH}_4\text{OH} + 8\text{NH}_4\text{NO}_3 + \text{O}_2 = 4(4\text{NH}_5.\\ \text{Cu}(\text{NO}_3)_2) + 4\text{KCy} + 4\text{NH}_4\text{ Cy} + 14\text{H}_5\text{O}. \end{array}$$

After an ammoniacal copper solution has been decolorized by a potassium cyanide solution it appears that chemical changes are going on in the solution for some considerable time (over four days) with the liberation of a free alkaline cyanide.

This may be explained by the breaking down of the cupric cyanides to cuprous; but to do so a more complex reaction is necessary than the one represented by the equation suggested by Mr. Sulman for the decolorization of an ammoniacal copper solution by potassium cyanide.

Mr. Brereton suggests that compounds of the x (NH₂) y (CuCy₂) z (Cu₂Cy₂) class are first formed, which pass to the x (KCy) y (CuCy₂) z (Cu₂Cy₂) class as the solution becomes colorless. Then if the latter class of colorless compounds takes some time to pass to its cuprous limit, namely, Cu₂Cy₂ 2KCN with the liberation of cyanogen, which in turn reacts with the ammonium hydrate, forming ammonium cyanide and cyanate, it accounts for the results observed.

Mr. Brown's paper is interesting on account of the practical information of the sulphuric acid process in comparison with the ammonia process described last year by Messrs. Jarman & Breteton

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The ore treated in which the copper is present as carbonate came principally from the Cobar Chesney Mine, Cobar, N. S. W., and was crushed in that company's old battery many years ago.

Separate acid vats were provided for the preliminary acid treatment. In the operations under review, these vats were

shallow, rectangular wooden vats, each holding about 25 tons of tailings. The cyanide vats held 75 tons of tailings, so that the contents of three acid vats were treated and subsequently loaded into one cyanide vat. On an acid vat being loaded, 10 to 12 tons of dilute H2SO4 solution were pumped on. As soon as the vat was full of solution, covering over the ore for some inches, the bottom valve was opened and the spent solution allowed to drain off slowly through a launder to the copper precipitation boxes. When the solution had drained so as to show the ore on top of the vat, the bottom valve was again closed for an hour or so and the acid allowed to remain in contact. This first solution when drained off never showed free acid. On again opening the bottom valve, the balance of the acid solution was allowed to percolate, usually at the rate of about 2 tons an hour, and flowed directly through the copper precipitation boxes to the sump, for making up again with H2SO4 or to waste.

The acid solution was followed immediately by a first-water wash, equal in tons to the original solution, which flowed into the vat from a tank above at the same rate as the vat was draining. This wash was followed by a clean water wash of about half the quantity, and the vat was allowed to drain for discharge, the total time of treatment being 48 hours. All the acid solution and wash passed through the copper precipitation boxes, and was either used over again or run to waste, depending on the supply of water available. The final wash was always clean water. When these operations were complete and the vat ready for discharge, 0.5 per cent to 0.8 per cent of lime was distributed over the surface and discharged with the sands, ensuring a fairly intimate mixture.

The copper was precipitated from solutions by passing through two boxes 10 feet x 3 feet x 4 feet, each divided into four compartments as in the ordinary zinc boxes. These boxes were filled with scrap sheet iron, obtained locally in the shape of old tins which had been burnt for recovery of the solder, and were to be had for the cost of carting. The precipitation of Cu on the large surface of iron thus exposed was practically perfect. On the other hand, all free acid entering the boxes was consumed at the expense of the iron.

Fifteen tons of weak KCy solution were first applied to displace the approximate 15 tons brought as moisture from the acid vats. This first 15 tons carried little gold. As a rule, about 12 tons would come through carrying only traces, and the other three tons would vary according to the relative perfection of the displacement. In any case, the first \$5 tons were carefully isolated in a special sump, and eventually used as a final wash for the outgoing residues. This solution passed slowly through a zinc box, and then through about 20 cubic feet of packed charcoal. The zinc precipitation was not effective unless free KCy was present, showing defective displacement in the vat, in which case there would be high gold values in the solution, but when no free KCy was present and but little gold, the charcoal always caught some, if not all. If assays showed that more than a few grains of gold were still present, and precipitation had been imperfect, the solution was again passed through a freshly made-up zinc box, with addition of KCy, before being finally used up as a wash.

From the 450 tons treated during this period 167.7 ounces of gold are theoretically shown as recovered. The actual recovery over a period of nine months was about 1 per cent in excess. The average value of the sands treated is shown as 9 dwt. 4 gr., of which 7 dwt. 11 gr. is recovered, leaving 1 dwt. 17 gr. in the residues, equal to an extraction of 81.1 per cent. The average theoretical strength of acid required per ton is shown as 0.65 per cent H₂SO₄, equal to 15.6 pounds, of 92 per cent acid. The theoretical quantity of acid called for is 6,588 pounds, a difference of 1 pound per ton coming in on correction of tonnage treated. The quantity consumed is 9,057 pounds of 92 per cent H₂SO₄, equal to 20.1 pounds per ton treated, or 4.5 pounds in excess of the actual requirement per ton of sands. The copper present averages 0.32 per cent, or

equal to 1.44 tons for 450 tons, 7.1 pounds per ton. Cyanide consumption is shown as .072 per cent, or 1.6 pound per ton.

From costs taken over several months:

3363 60	iken o	101 3		441			 				P	er Tor
Scrap	iron	aver	iged	1.				 0				1.5d
Lime	avera	ged			0	 0	 0		0	0	0	3.6d
Zinc	avera	ged			0		 0	 ۰		0	0	2.0d

The quantity of acid used—4.5 pounds per ton in excess of consumption on the ore—is open to criticism, and could to a great extent have been obviated by a better arrangement. Instead of passing the acid solution direct to the copper precipitation boxes, where the excess of free acid was consumed by the iron, it might have been sent to a sump and restrengthened for a second application, or used as a preliminary wash on the next vat, and only passed through the precipitation boxes when showing no free acid. Under the circumstances in which the work was performed it was considered economical to sacrifice acid and iron for other considerations.

The cement copper recovered was readily disposed of, although, perhaps, at a low price, since the total tonnage was small. Shipments averaged about 60 per cent Cu, the balance

being iron and silica.

No serious difficulty was found in getting a reasonable extraction of the gold from the acid-treated sands. The first experiment showed a possible extraction of only 65 per cent, with six days' treatment, but this was found to be due to lack of aeration. While experimenting with various oxidizers it was discovered that sufficient aeration could be obtained by applying the solutions as before described. In practice the solutions were well circulated in the sumps, and the usual provision made of allowing the pump suction to draw a little air.

The working solutions seldom showed over 0.3 per cent Cu, and usually were much lower. When, as latterly, more slimes had to be treated in the charges, conditions were not so favorable, and more Cu had to be dealt with in the cyanide treatment. The zinc boxes then required careful attention, and the endeavor was to get as clean a precipitation of copper, as well as of gold, as was possible. Lead acetate was freely used, and the boxes freshly made up at close intervals. No cyanide solutions were at any time discarded. When the solutions were foul they were always high in gold contents, and as the precipitation of the gold was accompanied by precipitation of the copper, re-aeration restored their working usefulness.

THE CAPITALISTIC SIDE OF BRITISH ELECTROCHEMISTRY AND ELECTROMETALLURGY.

Last April the present writer reviewed the capital invested in, and interest on, the British electrochemical industries from the unclassified information thereon contained in the indispensable (in this direction) Garcke's Manual of Electrical Undertakings. Dividing these into the same four groups as last year the changes are as follow:

(a) Makers of electrolytic alkali and bleaching powder—The share and debenture capital of the Castner-Kellner Alkali Co. now stand at £675,000, some £25,000 of debenture having been redeemed. A dividend of 4 per cent was again paid last year. The Electrolytic Alkali Co., whose share capital amounts to £302,000, and debenture capital at £20,850, again paid no dividends on their ordinary shares, but owing to an improvement in trade conditions were able to pay part of the arrears of dividends on their preference shares.

(b) Makers of primary and secondary batteries—Taking only these firms which devote their sole attention to this matter, the aggregate capital is about £622,800, the number of firms being sixteen. The Chloride Electrical Storage Co. again paid 8 per cent on their ordinary share capital; also a debenture debt of £20,000 has since been redeemed. The Electrical Power Storage Co. also paid the same dividend—6 per cent—on their ordinary shares as last year. The D. P. Battery Co. again paid 20 per cent on their small ordinary share capital of £10,000. The Hart Accumulator Co. paid 12½ per cent on their £40,000 of ordinary share capital, 6 per cent on their preference capital of £3,000.

(c) Electrometallurgical undertakings—The aggregate capital invested in these is £2,117,000. A decline due to the British Aluminium Co. having written off £90,000 of share capital is balanced by increases in other directions. In regard to this section it is rather difficult to define the difference between it and section (d), which embraces a number of smaller firms, the work of some of which is partly metallurgical.

(d) Miscellaneous undertakings number eleven, and have an aggregate capital of £444,000, there being no notable change since last year.

MARKET QUOTATIONS DURING MARCH.

The chief feature of the month has been the high prices reached by both tin and copper. The former fell from £165 to £162 between the first and fifth, and then rose to £169 by the thirtieth, and has since gone higher. Copper, which opened at £78.5, has twice touched £84, and closed only a fraction easier at £83.15.

In regard to iron, there have been no marked fluctuations, Cleveland warrants having closed at £2.8.7½. Lead has been easier, and has closed at £16 for English ingot. Zinc sheet is fetching £28.15 per ton. Platinum is quoted at £4.11.6 per ounce, at which price dealings are very much restricted.

As to chemicals, nothing calls for comment save that copper sulphate has advanced to £25 per ton. There is a marked all-round activity, which is somewhat hampered by the high price of copper.

LONDON, April 7.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

ELECTROCHEMISTRY.

Calcium Cyanamide.—The April issue of Harper's Monthly Magazine contains a well-written and interesting summary, by R. K. Duncan, of the work which has been done in recent years towards the fixation of atmospheric nitrogen for the production of fertilizers. The whole problem has been discussed repeatedly in our columns and at length in our last issue. Prof. Duncan's article contains, however, several new incidental notes. A new use of cyanamide is mentioned, namely, its employment for the case-hardening of steel, and

it is stated Loewe & Co. continually use large quantities of cyanamide in the manufacture of tools and of arms for the German Government. By the action of acids upon calcium cyanamide one obtains dicyandiamide,, which is now made by the ton, "and much of it is sold to the dye industries for a purpose that cannot be imagined by the manufacturers. Still, other quantities are sold to manufacturers of explosives, owing to the fact that when mixed with other substances it lowers the temperature in the gun-barrel." By reaction of cyanamide with water artificial urea is produced, tons of which have been

sold to manufacturers of pharmaceutical preparations. Finally, sarcosin unites with cyanamide to yield creatine—one of the actual substances of human muscle found in the extract of meat. The author makes the following prophecy: "We may look forward with a very reasonable assurance to the creation of as many factories for the fixation of elemental nitrogen as we have smelting furnaces for unfixing elemental iron."

Copper Refining.—In the Stockholm Teknisk Tidskrift, B. Carlson, who visited this country somewhat over a year ago, discusses the electrolytic refining of copper in the United States. The description of the various refineries contains nothing new, but the following notes contain some information which, although well known to refiners, does not appear to have been published before. An English translation of the whole article may be found in The Mining Journal (London) of Jan. 20:

Of the impurities of the anode copper, gold and platinum pass entirely over in the slime; only a small quantity of silver (7-10 grams per 1,000 kilograms) pass over to the cathodes, when the composition of the electrolyte remains normal. Considerably larger quantities of silver are found in the cathodes when the acidity of the electrolyte diminishes.

To guard against the loss of silver, a small quantity of hydrochloric acid (0.003 to 0.006 per cent of Cl), is added to the electrolyte when the water employed is not already sufficiently acid. If the addition of hydrochloric acid is too great, a danger of short circuiting arises, owing to the cathode copper depositing irregularly and forming needles on the cathode. The chlorine is recovered from the slime as chloride of silver. The amount of arsenic in cathode copper is not allowed to exceed 0.co1 to 0.002 per cent. In the anodes arsenic occurs partly as metal and partly as copper arsenite, which remains insoluble in the anode slime, and only for a small part is made soluble by secondary processes. Metallic arsenic is dissolved as arsenic acid, and is not deposited on the cathodes as long as the solution is sufficiently acid and not saturated. Experiments have been made to obtain refined copper, absolutely free from arsenic, by adding up to 20 per cent sulphate of ammonia. A frequent renewal of the electrolyte is, however, more practical. The above mentioned slight addition of hydrochloric acid, in presence of arsenic and also of antimony, advantageously hinders the tendency of these metals to deposit on the cathodes at higher densities.

The small addition of hydrochloric acid is one of the most important advances made in the electrochemical refining of copper; by this the now usual employment of high-current densities is rendered possible. The antimony passes completely in the solution, but is not deposited so long as the electrolyte remains normal. Selenium and tellurium, as far as is known, exercise no influence worth noting on the cathode copper, and lead passes entirely with the anode slime. Nickel remains in the form of sulphate in the solution. When the acidity of the electrolyte is too low, or when the current density is too small, the cathode copper is often so brittle that it can be pulverized. This particularity is caused by the high percentage of protoxide of copper in the deposited copper. The impure electrolyte removed from the baths is utilized for copper sulphate. Its free acid is neutralized by treating with copper granules and air, whereby the greater part of the ferric sulphate is deposited as ferrous sulphate. The solution is repeatedly boiled until, on cooling, copper sulphate crystallizes from the remaining liquor. Copper is separated with iron, and the liquid is run off. At chrome the electrolyte is only partly neutralized, and during crystallization receives a strong sulphuric acid solution. This is again heated, and sulphate of iron, nickel and copper are precipitated. The warm acids are filtered and allowed to cool, and arsenic acid crystallizes in large quantities. At the same time antimony antimonic acid are precipitated. The acid is used for treating the anode slime.

As is well known, the electrolyte must be kept in circulation,

otherwise copper and sulphuric acid weaken at the cathodes, and the impurities are deposited on the cathodes. The circulation is obtained by allowing the electrolyte to flow from an elevated vat, in which it is heated to nearly 50°, to the baths, which are placed one after the other, and it is again pumped back. Blowing air into the electrolyte at definite intervals aids the deposit of impurities, and suspended bodies coagulate at the bottom of the bath in colloidal solutions. Otherwise these suspended impurities deposit mechanically at the cathodes. The anode slimes consist of 10-40 per cent partly free copper and partly oxide, arsenic and sulphur compounds, 5-15 per cent silver, and 0.01-0.7 per cent gold; to this must be added small proportions of iron, bismuth, lead, antimony and arsenic compounds. As a rule the slime is treated in a phosphor bronze centrifugal, where the larger grains of copper, which represent about 10 per cent of the slime, are separated and then taken to the anode furnaces. The mother liquor is led back to the electrolytic department, and the slime is mixed with 95 per cent sulphuric acid at 66° and heated by steam, during 12-14 hours in lead-lined vats. By this process two-thirds of the arsenic, all the iron and bismuth, and a large part of the copper are dissolved. After settlement of the residues the solution is pumped to the vitriol works. The slime is boiled with water; this causes the precipitation of the copper and silver present from the sulphate of silver in the solution. If the slime does not contain sufficient copper, small quantities of raw slime are added.

The remaining slime, rich in precious metals, is first decanted and then filter-pressed, and dried in cast-iron vessels over a furnace. Silver containg 1-2 per cent gold and about 2-3 per cent copper is recovered by smelting in reverberatory or muffle furnaces with sand and soda. The slag, containing 1 per cent silver and 8-10 per cent copper, goes back to the raw copper furnace. The silver is refined, either by the electrolytic process, or treated in the ordinary way with concentrated sulphuric acid, and worked up to silver 0.998-0.9985 purity and gold 0.933-0.996 purity.

Electrodeposition of Copper Upon Iron.-In the January issue of the Journal of Physical Chemistry, O. W. Brown and F. C. Mathers give an account of a long series of experiments which lead them to recommend the following bath: Sixty grams copper sulphate, 50 grams sodium hydroxide, 159 grams sodium potassium tartrate, 1,000 cc. water. The best working conditions for this bath are a current density at the cathode of 0.1 to 0.5 amps. per square decimeter and a maximum density per square decimeter at the anode of not over 1.04 amps. If at any time a green precipitate should begin to form on the anode, a little caustic soda-3 or 4 grams per liter-should be added. The substitution of potassium hydroxide for sodium hydroxide and the warming of the bath were found to be detrimental to its working. Rolled, electrolytic and cast copper anodes work equally well. In all quantitative experiments of the authors, the anode corrosion had the theoretical amperehour efficiency if the anode discharge potential had a value of - 0.3 volt or less, but whenever the value changed to - 0.6 volt or greater, the anode corrosion became very poor. The tarfrate baths are superior to the cyanide baths, in that they are non-poisonous, do not give off an offensive gas during electrolysis, will work at a higher anode density, give as good cathode deposits, require a lower voltage and yield a current efficiency of 100 per cent at both electrodes. The bath requires the addition of small amounts of caustic soda from time to time. However, it is not inferior to the cyanide bath in this respect, as the latter requires the addition of potassium

Regulating the Voltage at the Terminals of Electric Furnaces.—The April issue of the *Electric Journal* contains an interesting illustrated article, by H. R. Stuart, on potential regulation for large electric furnaces. The article discusses in

detail the design of regulators made by the Westinghouse Co., which were already described in an article of FitzGerald in our Vol. III., p. 9, and following.

Distillation of Metals.-H. Moissan has recently made some interesting experiments which are described in the Comptes Rendus of March 19, and abstracted in the London Electrician, April 6. These experiments relate to the liquefaction and distillation of metals. The iron family require considerable energy in the electric furnace, but even tungsten and molybdenum may be regularly distilled. Boron and carbon, on the other hand, pass direct from the solid to the gaseous state. Brought to the temperature at which carbon evaporates, titanium is liquefied. The author has distilled 17 grammes of titanium in 6 minutes. A difficulty lies in the avidity with which titanium combines with nitrogen. The author draws some conclusions with regard to the temperature of the sun, which is known to contain titanium. "On account of the large amount of heat radiated by the sun, it appears probable that it contains some solid matter as well as gaseous matter. At atmospheric pressure, no solid matter can exist at temperatures superior to that of the electric arc; i. e., 3,500°. Under the pressures available within the body of the sun there will be much solidification, so that bodies can remain solid at temperatures higher than that of the arc. It follows that the temperature may be higher, and that at the surface of the sun it may amount to the 6,590° given by Wilson."

Blast Furnace Gas for Electrolytic Work.—In the London Electrical Review, of March 23, W. H. Booth refers to the fact that the large electrochemical industries have so far looked for cheap water-power, and that for this reason Niagara Falls and the upper Savoie have become centers of electrochemical activity. The author points out—what has been often emphasized in our columns—that for certain electrochemical and electrometallurgical work the blast furnace, running as it does 24 hours per day, may become a valuable power producer, the blast furnace gases, of course, to be used in gas engines coupled to electric generators. The author remarks that a British aluminium company hopes soon to have 30,000 hp. at work, an amount of power that would be obtained from less than 2 per cent of the blast furnaces of Great Britain.

METALLURGY. IRON AND STEEL.

Properties of Cast Iron.-Dr. Borcher's Metallurgie, having now extended its field to cover "die gesamte Hüttenkunde," and Dr. F. Wüst, of Aachen, having taken over the assistant editorship of the journal, with special oversight of iron and steel, the first number under the new regime, Jan. 8, contains very appropriately a long communication from Dr. Wüst himself, "mit 3, Abbildungen, 8 Kurven und 4 Tafeln." There are really some good points brought out in the article. Dr. Wüst investigated twenty varieties of cast iron, of each of which careful chemical analyses were made for graphite, carbide carbon, hardening carbon, silicon, manganese, phosphorus and sulphur; in almost every case the iron and carbon amounted to 99.9 per cent, the carbon varying from 2.56 to 4.82 per cent. With these alloys of iron and carbon very carefully-made cooling curves were obtained. They showed, in general, melting points from 1.112° to 1.149° C., independently of the per cent of carbon between 2.94 and 4.66. The conclusion is drawn that with 3 per cent and over of carbon a minimum melting point of about 1,130° is reached. Between 1,130° and 700° the cooling curve falls regularly, but at 700° a sudden evolution of heat takes places, corresponding to the temperature at which iron carbide is insoluble in iron, the whole mass settling into an intimate mixture of cementite and ferrite crystals. The change is visible in both gray and white iron, in amount corresponding to the quantities of perlite contained in each. The change takes place between 692° and 719°, but no law connecting these irregularities with the composition was discovered.

Phosphorus and Carbon.—Fettiveiss, in the metallurgical laboratory at Aachen (Metallurgie, Jan. 22) has reinvestigated the influence of phosphorus on the dissolving power of iron for carbon, in order to control Stead's researches of 1900. He improved on Stead by taking a pure highly carburized iron and adding to it ferrophosphorus, thus eliminating excess of insoluble carbon, and by analyzing the iron for phosphorus after the tests; whereas Stead added phosphorus to pure iron and then tried to saturate it with carbon, and did not control the test by subsequent analysis. The results are not far from Stead's, showing a steady decrease of 0.5 per cent of carbon for each 2 per cent of phosphorus present, and confirming the supposition that the phosphorus forms Fe³P, containing 15.8 per cent phosphorus, and that since this can hold only about 0.25 per cent of carbon, the total carbon present must steadily decrease to nearly zero as the phosphorus approaches 15.8 per

Manganese Alloys.—Levin and Tamman (Zeitschrift für Anorgonische Chemie, 47, 136—from Metallurgie, Jan. 22) have made a much needed investigation of the melting points of iron-manganese alloys. Using soft steel with 0.26 per cent impurities, melting at 1,550°, and 99.4 per cent manganese, melting at 1,245°, the melting points were:

10	per cent	manganese			*	×	*	*				*	1,525°
20	66	46				*							1,483°
30	6.6	46											1,442°
40	44	44				×			*				1,380°
50	**	44		,					*	×			1,340°
60	44	44				*			×				1,288°
70	**	44	 *			*			*		,		1,275°
80	**	**			×		÷		*	×			1,260°
90	**	66			×								1,250°

An inspection of the curve shows that the melting point of the steel is reduced pretty regularly 4°.4 for each per cent of manganese present, until 60 per cent is reached, and thereafter close to 1° for each additional per cent up to pure manganese. It must not be forgotten that these are the carbonless alloys.

Deoxidation of Calcium.—C. Quasebart, of Aachen, relates, in Metallurgie for Jan. 8, his fruitless endeavors to alloy calcium with iron. He melted soft steel in a plumbago crucible, tied a piece of calcium to a soft-iron rod, with iron wire, and dipped it into the melted steel; the carbon in the steel fell 0.37 per cent, but no calcium was left in it. Throwing pieces of calcium into an ingot mould, while pouring 0.87 per cent carbon steel into it, likewise produced no alloy. A piece of calcium was bored out, a round iron cylinder tightly pressed in, and the two heated together 60 hours at 750°; not a trace of calcium diffused into the iron. A soft iron cylinder, filled with pieces of calcium was heated 31/2 hours to 1,400°, but no calcium alloyed with the iron. A similar cartridge was put into melted cast iron with equally negative results. The investigator concludes that he has proved the universal negative, concerning the ability of iron and calcium to alloy with each other.

LEAD

Alloys with Arsenic .- Prof. K. Friedrich, of the old Freiberger Bergakademie, is continuing his good work on the metallic alloys. Jan. 22 number of Metallurgie contains a tenpage description of the combinations of lead with arsenic in increasing amounts up to 34 per cent, with nine interesting photomicrographs. Pure lead, melting at 327°, has its melting point reduced by arsenic until the 2.5 per cent arsenic alloy melts at 292°, forming a well-marked eutectic, which appears also in all the other alloys investigated. Above 3 per cent arsenic the melting point rises regularly, becoming 563 at 34.4 per cent, with no intermediate indication whatever of any compounds such as Pb2As, Pb2As2, Pb4As2, PbAs, or Pb3As4, which have been spoken of by Descamps. The latter must have had only mixtures. Alloys of lead and arsenic may exist above 34 per cent arsenic, but as far as can be predicted it is not likely.

COPPER.

Pyritic Smelting.-A paper by R. C. Alabaster and F. H. Wintle, before the Institute of Mining and Metallurgy, gives an account of the practice of pyritic smelting at the Tennessee Copper Co. and at the Ducktown Sulphur, Copper & Iron Co., at Ducktown, Tenn. The two-stage process is employed, namely, the ore is first concentrated to a matte giving 10 per cent of copper, and this matte is resmelted in another furnace to a matte containing about 45 per cent of copper. The enriched matte is bessemerized. In the concentration furnace 280 to 300 short tons of the 10 per cent matte produced in the first furnace are smelted every 24 hours. The Ducktown Sulphur, Copper & Iron Co. also employs the two-stage process. In the first stage there is made a matte containing 20 per cent of copper, the slag running 0.37 per cent in copper and containing 32 per cent of silica, 38 per cent iron and 8 per cent lime. The matte is concentrated to 50 per cent copper, the slag containing 0.6 per cent copper. The coke consumption is 8 per cent of the matte.

The authors think that for the treatment of an ore of the nature of the Ducktown ore the two-stage process is the most advantageous, and they recommend three or more rectangular water-jacketed blast furnaces, approximately 200 to 240 inches long, and not more than 44 inches or less than 40 inches wide. Wider furnaces have a tendency to run cold and form crusts. Good results would be obtained by using 20,000 cubic feet of free air at 30 to 35-ounce pressure with an ore column 12 feet high. The first concentration should produce a matte carrying 10 per cent copper and a slag with not more than 0.25 per cent copper. It is necessary to practice low concentration in this stage so as to keep down the slag losses. The matte should be treated in a second furnace with the necessary amount of silica for fluxing off the iron. The slags, containing about 0.6 to 1 per cent of copper, should be retreated in the first furnace. The composition of the slag to be formed depends upon special conditions, but the aim should be to produce an ideal slag of a mixture of bisilicate of lime and singulosilicate of iron in equal parts. Alumina is the only substance which interferes with pyritic smelting, since it forms a most infusible silicate and necessitates a higher consumption of fuel. Zinc, if its percentage is below 6 per cent in the ore, does not interfere with the process. The authors think that hot blast is unnecessary when the sulphur in the ore exceeds 20 per cent. Where the sulphur is below that percentage and coke is expensive, while other fuels, such as wood, oil, gas, etc., are comparatively cheap, hot blast may probably be employed with advantage. For all ordinary propositions, however, the authors unhesitatingly recommend the use of cold

GOLD.

Fine Grinding and Cyanide Process.—A paper by F. C. Brown, published in the January issue of the Bi-Monthly Bulletin of the American Institute of Mining Engineers, gives a review of the cyanide treatment of ores in the Ohinemuri district of New Zealand. According to the author, extremely fine grinding (so that about 90 per cent will pass a 200-mesh screen) will become a very important feature in this district in the near future. A review of present practice is given The author shows the following advantages arising from the system of very fine grinding in tube mills when compared with coarser crushing. Firstly, the dissolving of the gold and silver in the ore takes less time; secondly, weaker cyanide solutions can be used, and, thirdly, a better extraction can be obtained. From many experiments in treating concentrates by the cyanide process the author has found that the following points are of great importance: The concentrates must not be allowed to become decomposed by being left exposed to the air. Exceedingly fine grinding is necessary in most cases, and usually the concentrates must be reduced to slimes. Agitation must provide for a good supply of air. Weak cyanide solutions will

give a good extraction, provided the grinding is fine enough. Since all these requirements would be obtained by treating the mineral particles; i. e., the concentrates, together with the sand, it would appear as if the special operation of concentration could be dispensed with in some cases, which would result in a great saving in cost of plant and operating expenses.

ALLOYS.

Aluminium-Magnesium.-Louguinine and Schukareff publish in Revue de Metallurgie for January, a very carefully prepared article on the heat of combination of these two metals to form alloys. They tested alloys corresponding to the formulæ, AlMg, AlaMg, AlaMg, AlMg, AlMg, by dissolving them in hydrochloric acid in a calorimeter, and comparing the heat evolved with that of the solution of equal quantities of aluminium and magnesium unalloyed. In no case could they detect any differences, showing that the two metals alloy without evolution or absorption of heat. The two data which remain as the useful result of their work, are the heats of solution of aluminium and magnesium in HCl.10H3O. These are 4,612 calories per gram of aluminium, and 4,510.9 calories per gram of magnesium, corresponding to 124,524 per atom of Al. and 108,262 per atom of Mg., which represent 41,508 calories and 54,131 calories per chemical equivalent of the respective metals.

Analysis of Current Electrochemical Patents

ELECTRIC FURNACES.

Pressure Furnace.—H. N. Potter, 814,727, March 13. Application filed July 23, 1903.

Dr. Potter, who has done a great deal in the past in working out the design of electric tube furnaces, patents here a tube furnace for use in such processes in which gases or vapors are to be heated to a high temperature and to be subjected simultaneously to a high pressure. The ends of the heating tube are clamped between terminals of carbon which are connected to the outside cylindrical wall of the furnace. This is made in two tubular parts, separated by an insulating gasket. The space between the outside wall and the inner heating tube is filled with an insulating refractory packing material. The current passes from one-half of the outside wall (which forms one pole) to the carbon terminal at that end, then through the inner tube to the carbon terminal at the other end, and leaves the furnace through the second half of the outside wall. For details of the design, which is very carefully worked out, the reader must be referred to the patent specification.

Vacuum Furnace.—H. N. Potter, 814,726, March 13, Application filed July 23, 1903.

The purpose of this furnace is just opposite to that of the type described in the preceding abstract. It is designed for highly heating refractory materials under reduced pressure, or for executing certain high-temperature chemical reactions, where the speed of the reaction or its efficiency is improved by reduced pressure. The material to be treated is packed between the water-jacketed outside walls of the furnace, and a tube in the center of the furnace and the air between the particles to be heated is exhausted by means of a pump through the central tube or otherwise. This furnace, like the one in the preceding abstract, is of highly finished mechanical design, and many details of the construction are quite elaborate.

ELECTROLYTIC PROCESSES.

Iron from Ore,—R. H. Aiken, 816,142, March 27. Application filed June 1, 1903.

Pulverized Fe₂O₂ or Fe₃O₄ is gradually fed into a molten bath of FeOSiO₂, with an addition of CaO, MgO, and the

bath is electrolyzed, the precipitated iron serving as cathode. As much as 20 per cent of Fe₂O₄ can be dissolved in a basic silicate of proper constitution. The process is continuous, and the cell may be made large.

Aromatic Alcohols and Derivatives.—C. Mettler, 815,548, March 20. Application filed Nov. 2, 1904.

Aromatic alcohols and their ethers are produced by "exposing the aromatic esters in a dissolving agent, capable of conducting the electric current in the cathode space of an electrolytic apparatus, using for the cathode a material of specially high cathodic tension, then neutralizing the cathode liquid, then separating the reaction product and finally distilling." The following is an example: The cathode space of ' a cell is charged with a solution containing 300 grams of ethyl ester of benzoic acid, 350 grams of concentrated sulphuric acid, 500 grams of alcohol of 96 per cent, 100 grams of water. (In place of water a correspondingly dilute solution of sulphuric acid can, of course, be employed.) The anode space, separated by an earthenware cell, contains sulphuric acid diluted with water. It is known that the electrolytic reduction of substances, which are not readily and easily reduced, can be facilitated by employing for the cathode a metal having a high cathodic tension. Therefore, the cathode is made of pure sheet-lead, which is suitably prepared according to Tafel's prescription (Ber. d. Deutschen Chem. Ges., 33; 2,215), and pure lead is employed for the anode. A current of 7 amps. per 100 square centimeters cathode-surface is sent through the cell; but the strength of the current can be materially varied without disadvantage. By means of cooling, the temperature is kept between 20° and 30° C. When no more hydrogen is absorbed, the cathode liquid is neutralized with about 30 per cent soda-lye. Two layers are formed, of which the upper one consists of alcohol and the products of reduction of the ethyl ester of benzoic acid. This is separated, distilled and fractionized. The benzyl-ethyl ether boils at 184°, the benzyl alcohol at 205°.

Aromatic Alcohols.—C. Mettler, 815,193, March 13. Application filed Oct. 11, 1905. (Assigned to Badische Anilin und Soda Fabrik.)

With reference to the process, described in the preceding abstract, the inventor states that the free carboxylic acids themselves can also be reduced in an analogous manner, giving rise solely to the alcohols. It is necessary to employ a cathode of a metal of high cathodic tension like lead. The course of the reaction is illustrated by the equation CoHs. COOH + 4H = C₈H₅. CH₂OH + H₂O. One example given is as follows: Introduce into the cathode compartment of an electrolytic apparatus a solution of 200 parts of benzoic acid in 400 parts of concentrated sulphuric acid and 1,400 parts of alcohol. The cathode consists of sheet-lead, prepared according to Tafel. The anode consists of pure lead, and the anode compartment is filled with dilute sulphuric acid. Send through the liquid an electric current of from 6 to 12 amps. per 100 square centimeters of cathode surface and cool by means of water, so that the temperature remains between 20° and 40° C. As soon as no more hydrogen is absorbed neutralize the cathode liquid with caustic soda and extract the alcohol, which is formed by means of ether. The yield of benzyl alcohol boiling at a temperature of 201° centigrade is almost quantitative.

Production of Ammonia.—J. A. Lyons and E. C. Broadwell, 816,928, April 3. Application filed Sept. 3, 1904.

A crucible forming the cathode contains a fused bath of a borate of an electropositive metal (potassium, sodium, manganese, chromium, molybdenum, tungsten, vanadium, etc.) or a mixture of such borates, with a carbon rod as anode in the center. On account of the relatively small surface of the anode rod, the anodic current density is high and an intense heat is created at the carbon anode. The anode is surrounded by a pipe suspended into the fused bath, and through this pipe

nitrogen or a nitrogen-bearing gas is passed into the bath. "The electrolysis produces anion, boric anhydride and oxygen at the anode, where the intensely-heated carbon acts as a reducing agent to chemically reduce the anion to boron. The pipe serves to prevent the boron thus formed from floating over to the cathode, and the nitrogen introduced into the pipe combines with the boron to form boron nitride. The metal of the borates will be deposited at the cathode." After sufficient boron nitride has accumulated it is subjected, while separated from the metals, to steam at a temperature of 600° C., or above, when the reaction takes place $2BN + 3H_2O = B_2O_8 + 2NH_2$. The temperature of the bath is maintained at about $1,000^{\circ}$ C.

Purifying Water.—H. C. Bailey, 814,764, March 13. Application filed May 23, 1901.

A casing with insulating cover contains a plurality of spirally-arranged metallic strips or plates between the bottom and the cover. The inner terminals of these spiral strips are arranged near the center of the receptacle, while the outer ends extend to the sides of the receptacle. Channels are thus formed between the spiral strips. Water is passed through these channels in a rotary movement toward the center, where it enters a secondary channel in the reverse direction, flowing toward the periphery of the receptacle to an outlet opening. On the passage through the channels the water is electrolyzed, one side of the channel forming the anode, the other the cathode.

Electroplating Apparatus.—Guy L. Meaker, 815,027, March 13. Application filed Dec. 23, 1904.

The apparatus is intended for plating small articles. These articles rest on a horizontal tray with an open, foraminous or reticulated metallic bottom, which forms the cathode. Immersed within the electrolyte above the tray are the anodes, consisting, of course, of the metal to be deposited. The tray is agitated and vibrated in such a manner that the articles on it are constantly shifted in one direction and simultaneously turned so as to be coated on all sides.

Electrolysis of Water.—W. F. M. McCarty, 814,155, March 6. Application filed Nov. 8, 1904.

Apparatus for carrying out the process mentioned on page 113 of our March issue. The electrolytic cell is divided into two compartments by means of a solid diaphragm, which is perforated, short glass tubes being inserted in each perforation.

BATTERIES.

Storage Battery.—F. Monterde, 814,064, March 6. Application filed July 24, 1905.

A cylindrical lead vessel forms one pole of the battery. Within it are provided a number of concentric open-ended cylindrical electrodes provided on both sides with Y-shaped pockets to contain the active material.

Battery.—E. W. Schneider, 816,384, March 27. Application filed Aug. 11, 1905.

In a battery of several dry cells one cell may be thrown into or out of the generating circuit of the other cells by means of an end-to-end reversal. The object is to use this special cell only when the others have become nearly exhausted. By being connected in series with the others it then serves to boost the voltage.

Storage Battery.—H. B. Hallock, 814,691, March 13. Application filed May 15, 1905.

Details of construction of a storage battery with two electrodes in horizontal position one above the other and separated from each other by means of a porous separator. The lower electrode is the positive pole element, and its grid fits upon the inside surface of the inclined sides at the bottom of the jar.

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Active Material for Storage Batteries,—E. L. Oppermann, 815,628, March 20. Application filed April 11, 1905.

In order to produce an exceedingly porous and at the same time hard and tenacious active material, the usual active oxide is mixed with animal hair or wool, ground or powdered so that the length of any individual hair is not more than one-sixteenth of an inch. The hair should form from ½ to 1½ per cent of the mixture.

RECENT METALLURGICAL PATENTS.

Treatment of Sulphide Ores with Chlorine.—C. E. Baker and A. W. Burwell (816,061, March 27) patent a process of treating sulphide ores which is analogous to the first step in the Swinburne-Ashcroft process (our Vol. I., p. 412). The crushed, dry sulphide ore is charged into a pebble mill "and subjected therein with or without the application of heat to chlorine, in quantity sufficient to combine with the metals only of the ore, thereby separating sulphur in a physical state which will depend upon the temperature." In the revolving drum the particles of ore are finely ground, thus exposing fresh reacting surfaces to the action of the chlorine. The liberated sulphur remains mixed with the gangue, and may be recovered by distillation, etc. At the low temperature requisite for the distillation of sulphur no appreciable amount of metallic chlorides is said to be reconverted into sulphides.

Gas Producer for Zinc Furnaces.—C. Ellis (816,973, April 3) points out that when gas producers for zinc furnaces are operated with steam, it is in practice impossible to completely decompose the steam, which results in an adulteration of the producer gas by water vapor. This steam tends to deaden combustion in the furnace, and is said to have a disintegrating effect on the retorts used in the distillation furnace. C. Ellis, therefore, proposes to generate practically dry combustible gas in a gas producer. in which a deep bed of coal-fire is brought to a state of incandescence by the air of a blast produced by a fan. The generated dry combustible gas is burnt around the retorts or muffles by admixture of air preheated from the waste products of combustion. The products of combustion from the furnace preferably pass through a regenerator of the continuous type. A portion of the products of combustion is diverted either before or after its passage through the regenerator, and is returned to the gas producer for the generation of combustible gas.

Converting Zinc Sulphate Solutions into Zinc Oxide.-C. E. Dewey (815,516, March 20) patents apparatus for this purpose. The process consists of two steps; the solution (in which a quantity of fine coal is placed) being first evaporated and then calcined. Accordingly, two independently-revoluble cylinders are provided, being mounted in axial alinement. Their adjacent extremities are open, and in such proximity to each other that the heat may readily pass from one to the other. In one cylinder the calcining operation, or the removal of the sulphuric acid from the zinc sulphate, is effected, while in the other the solution is evaporated to dryness. The calcining cylinder is located in proximity to a fire-box, from which the heat and products of combustion pass directly into one end of the rotary cylinder, while the heat and products of combustion pass from the calcining cylinder into the evaporating cylinder, and thence into a dust chamber located at the extremity of the evaporating cylinder remote from the calcining chamber.

Aluminium-Tungsten Steel.—S. Parfitt (818,044, April 17) patents the manufacture of a steel of low carbon content, which is claimed to have a high tensile strength and not to be affected in the presence of strong solutions of sulphuric acid and similar corrodents. The iron is melted in an open-hearth furnace and decarbonized in the usual way, so as to contain not more than

0.1 per cent of carbon, thereafter 0.5 per cent of aluminium and 0.3 per cent of powdered tungsten are added.

Articles of an Iron-Hydrogen Alloy,—G. W. Gesner has formerly patented several processes of making alloys of iron and hydrogen. He now (815,419, March 20) describes a process for agglomerating the powdered alloy into strong coherent bodies. The powdered alloy is moistened and compressed by hydraulic pressure into the body of desired form, which is then fixed at a temperature sufficient to cause the component particles to first or partially fuse together without melting down into a liquid mass. For instance, plates ½ inch thick are maintained at about 2,000° F. for 2 hours. "The alloy is a material of great value for many uses, such as the electrodes of electrochemical apparatus, chemical containers, electrical conductors," etc.

Purifying Blast Furnace Gas.—For the utilization of blast furnace gas in gas engines nothing is more important than its purification and removal of all dust. This is done according to Q. Bent (815,674, March 20) by centrifugal force in two chambers, through which the gas is passed in succession. Both chambers contain each a fan. The gas is passed first upwards through a smaller chamber, where, by means of the fan, it is diverted from its upward course and thrown outwardly toward the periphery, whereby the heavier dust particles are separated and escape through four openings into downward pipes. The gas with the lighter dust particles, not yet removed, passes onwardly into the upper large chamber, where the lighter dust particles are removed by means of a large fan.

Refractory Material for Electric Insulation.—D. M. Stewart (816,270, March 27) patents a highly refractory fireproof material, which is at the same time a good electric insulator and is unaffected by cold or moisture. The chief constituent is magnesium silicate or aluminium silicate, found in nature as steatite; 100 parts by weight of finely comminuted steatite and 20 parts of-water glass are mixed to a plastic mass of about the consistency of workable putty. The mixture is brought into the desired shape by strong pressure, and the water contained in it is expelled in a drying kiln. The article is finally hardened by subjecting it to a temperature of from 1,500° to 2,000° C. for 5 to 8 hours, depending upon the bulk of the mass. Laboratory tests with minute article have shown the best results to be obtained by a 5-minutes' exposure to a temperature of about 1,500° C.

BOOK REVIEW.

METALLURGICAL CALCULATIONS. By Joseph W. Richards, A. C., Ph. D., Professor of Metallurgy in Lehigh University. Part I: Introduction, Chemical and Metallurgical Principles, Problems in Combustion. New York: McGraw Publishing Co. 201 pages. Price \$2.00.

The problem of the modern engineer is to carry on industrial enterprises at a maximum efficiency. To attain such a worthy object accurate knowledge of the science that underlies all material phenomena is indispensable. In all the technological world there has been a constant but slow increase in this knowledge. Thus, we have seen the millwright evolved into the mechanical engineer and the old "bell-hanger" replaced by the college-trained electrical engineer. However, the progress in metallurgy has been slow from the rule-ofthumb man to the technical engineer, especially in the United States. We except, of course, a number of brilliant advanced thinkers. The chief reason for this slow development is the lack of accurate metallurgical data on which to base metallurgical calculations. The advance made has been due to empirical, manly and noble endeavor, but has, nevertheless, been inefficient and wasteful.

In the past five years the introduction of new pyrometers

and other recording instruments, together with the spread of knowledge of modern physical chemistry, has resulted in a partial remedy of this bad condition. At the psychological moment, Prof. Richards has collected, systematized and elucidated all the fundamental chemical, physical and mechanical principles as well as empirical new data together with the old. The result is the book under discussion.

Since the main part of the book is a reprint of the articles which appeared in this journal from March, 1905, to March, 1906, no review of its contents is needed at this place. As a valuable new addition the appendix should, however, be mentioned, containing the statements of twenty-three more problems from practice, without solutions, but with a statement of the final answers.

Judging from the enthusiasm with which the Metallurgical Calculations were received in serial form by our readers all over the world, we are sure that they will be even more welcome in the more convenient and very handsome book form, and will produce much good in the direction indicated by the author in the following words of his preface:

"If the rule-of-thumb is to be replaced in a metallurgical process by scientific operation, the change must be based on experiments, classification of results and calculations therefrom. The principles involved are physical, chemical and mechanical; the scientific metallurgist must master these, use them as tools, and overcome brute nature by their skillful employment.

"Every metallurgical problem is an exercise in pure logic and mathematical reasoning; the premises are observed facts—all that can be learned of the process by direct observation and measurement; the conclusions desired are everything which can be deduced from the premises by hook or by crook, by direct logical process or by inference. In this way data and information are obtained which cannot be directly observed or measured, and which are of the most essential value for thoroughly understanding the process."

PRODUCER GAS AND GAS PRODUCER. By Samuel S. Wyer, Me. E. New York: Engineering and Mining Journal. 296 pages. - Price \$4.00.

This most excellent treatise fills a large gap in technical literature which had previously been only partly closed by the smaller work of Sexton. The rather too numerous (28) chapters are devoted principally to physical and chemical laws, other commercial gases, classification of gas producers (intelligently done), use of steam (very practically handled), efficiency (rather too brief), descriptions of pressure producers, suction producers, by-product coke ovens, power plants and gas poisoning.

There is more general information on these subjects within the covers of this book than in any other that we know of; at the same time, we note the absence in many cases of specific data which would be valuable, such as, for instance, the quantity of tar, ammonia and benzene recovered, the approximate chemical composition of the tar and its calorific power. Such details might be added in many places, to the improvement of the value of the book to the practical man. The thermal and physical calculations are very clearly explained, but from the standpoint of the mechanical engineer; the chemist has many short-cuts which are absent from Mr. Wyer's work, which would facilitate greatly the calculations if introduced with proper explanations.

Altogether, the work can be highly recommended to students and practical men as the best treatise there is at present on this specific subject.

Practical Electrochemistry. By Bertram Blount, F. I. C. London: A. Constable & Co. Third impression. Second edition, revised and enlarged. Price \$3.25.

That this is the third impression of Mr. Blount's now well-

known work is the best evidence that, as the author states in his preface to the second edition, the book has been found acceptable by a small but "discriminating" public. As the title implies the book necessarily deals with a very wide range of subjects, embracing processes for electrolytically winning nearly every known metal; processes for separating and recovering new metallic products and finally a series of organic separations and combinations which the modern practice of electrolysis has brought into use.

The strong point of this book, to our mind, has always been the systematic application which the author makes throughout his work of electrothermal and thermochemical data and laws. This gives to the subject a most useful connected sequence and plan, and the author evidently feels the value of this foricbly, as he is very fond of impressing upon the reader that "there is no magical efficacy in electrolytic processes per se," but that the boundary line of their utility is found in their economical factors; a somewhat self-evident truth one would imagine at this time of day.

The book begins with a theoretical discussion of the principles of electrolysis, with special reference to the dissociation theory. This subject receives adequate, if not luminous, treatment, and the author's style is thoroughly readable.

Section II. deals with the winning of copper, lead, gold, silver, nickel, cobalt, antimony and zinc, all from their aqueous solutions. There is some slight defective arrangement here, several fusion processes for lead, zinc, etc., are treated, which do not properly belong to the section under the main classification adopted. The other orders of classification also appear to be purely arbitrary and leave something to be desired.

In this section (as also in other parts of the book) we find much matter that is out of date. Old matters, like Borchers numerous unworked or abortive designs being frequently rehashed. The principal additions of value are a short amount of the Tommasi and Salom lead processes, the Phœnix zinc process and a few minor matters, but no account at all is given of such actual and importnat operations of recent date as the Betts refining process now at work at Trail and St. Helens, and soon in South Chicago.

Section III. nominally deals with all igneous solution processes, but actually is confined to aluminium, magnesium and sodium. With the exception of a brief notice of a few recent developments, like Ashcroft's sodium process, we do not find that any fresh information has been produced here.

Section IVV. deals with the electric furnace proper in all its branches. This subject receives able treatment from the electrothermal side, and constitutes one of the striking values of the book. The statistical and practical account of the various industries, however, is somewhat weaker, and such important subjects as phosphorus receive altogether inadequate treatment.

The same remarks apply to Section V., which deals adequately with the recent developments in iron and steel smelting, and to Section VII., which treats of alkali and chlorine processes. Section VI., in our opinion, had been better omitted altogether. Section VIII., the electrolysis of organic compounds, is a valuable addition to the work, more especially as it has not been customary hitherto in electrolytic text-books to touch on this subject. Section IX, and last, deals with "power," and gives the author another opportunity to emphasize his favorite viewpoint of the thermochemical side of his subject.

Altogether an admirable text-book, and one well worth keeping up to date. It bears evidence of thoughtful and scholarly compilation. Its weak points are those nearly unavoidable in such cases, and arise from limited opportunities and limited experience in the "works" direction. Such limitations are often almost inseparable from the lot of the professor or laboratory man. We cordially welcome and praise Mr. Blount's latest effort.

Reduction Plant of the Vanadium Alloys Company.

The uses of vanadium were formerly quite restricted, vanadic acid being employed chiefly in calico printing, in the manufacture of aniline black and in the coloring of glass. But

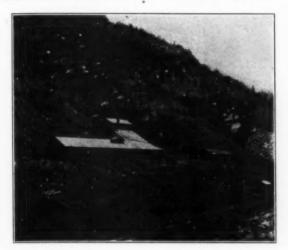


FIG. I.-REDUCTION PLANT OF VANADIUM ALLOYS CO.

this rare element achieved an eminent practical importance when it was found that as an addition to steel it is able to give it certain extremely valuable properties.⁴

It is well known that the revolution in workshop practice, due to the introduction of high-speed tool steels only a few years ago, has brought about a new and highly important industry—the manufacture of ferroalloys. This commercial evolution has been the subject of numerous articles in our



FIG. 2.-ROLLERS.

columns during the past years, since it represented one of the most interesting current electrometallurgical developments. It is, of course, well known that in the case of highly refractory oxides and for the production of high percentage alloys, the electric furnace is indispensable.

In this development Europe has formerly been leading, although in the Willson Aluminum Co. this country has the pioneer producer of ferrochrome.

Ferrovan a d i u m has been produced

in the past only in small quantities; none whatever on a commercial scale in this country. Most of the ore that has been taken out in the past from the deposits in Western Colorado was shipped to Paris, and there worked up into the ferroalloy.

It is with great pleasure, therefore, that we can record in this article the successful starting of a ferrovanadium plant on Colorado soil. It is the first mill of its kind, built not only

¹An excellent summary of researches in this field was given by Dr. Hans Goldschmidt, in our Vol. III., page 169.

in America but in the world, and credit must be given to Mr. Tyndal Rynard, of New York, for his untiring efforts in building this first commercial plant, whereby the value of vanadium can be demonstrated on a large scale as an alloy for steel. The general offices of the Vanadium Alloys Co. are at 25 Broad Street, New York City. The reduction works are located at Newmire, Col. Mr. Tyndal Rynard, who is a well-known member of the American Electrochemical Society, is the general manager of the company. Dr. Warlimont, late assistant to Dr. Borchers at the Institute of Technology, at Aachen, Germany, has charge of the splendidly equipped works laboratory. The present capacity of the reduction works is 40 tons of ore per day.

The ore worked in the mill runs from 2.5 to 6 per cent of vanadium. The supply of vanadiferous sandstone at the disposal of the company is almost inexhaustible. The company controls mines covering a territory along the San Miguel River of over 12 miles. For descriptions of it reference may be made to the report of Dr. W. F. Hildebrand on vanadiferous minerals in the American Journal of Science, August, 1900.

Fig. 1 gives an outside view of the mill at Newmire, Col. Fig. 3 shows one of many mines, 2 miles from the mill. The illustration shows the limestone cap above the ore, which occurs in the form of a blanket vein about 9 feet high.

The process consists essentially of three steps. The first is crushing and roasting with a salt.

The second step is leaching with water and precipitating, by means of ferrous sulphate (green vitriol), a vanadate of iron which contains 50 to 75 per cent V₂O₅, and is free from silica and other impurities.

The third step is the reduction of this vanadate of iron in the electric furnace. The final product of the process, as carried out by the Vanadium Alloys Co., is ferrovanadium containing 25 to 50 per cent vanadium with less than 1 per cent of carbon.

The crushing, roasting and leaching plant was designed,



FIG. 3.-MINE.

built and equipped by the Traylor Engineering Co., of New York City. Although the first works of its kind, the mill was so carefully planned and constructed that as soon as started a few weeks ago it worked to perfect satisfaction. The adjoining illustrations show several interesting details of the plant. For the original photographs we are obliged to Mr. Tyndal Rynard and to the Traylor Engineering Co.

Fig. 2 shows the rollers in the crushing mill. There are three hoppers shown, one for the ore, the second for the salt, and

the third for the oversize from the screens. The first and the second hopper are worked intermittently. The Traylor centripact screen, which was described in our last volume, pages

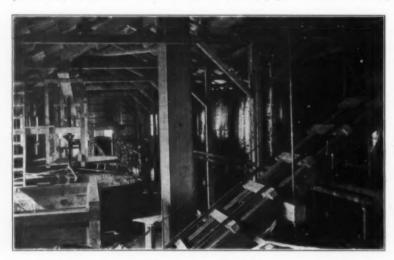


FIG. 4.-LEACHING PLANT.

124 and 163, has here again proven a most valuable piece of apparatus.

After being crushed and roasted the ore is brought to the cooling floor, which slopes both ways to a line, which is inclined downwards to a box discharging into the elevator. The roasted ore is dumped on the brick cooling floor, and then by means of a hose connected with a pump, the material is washed into the elevator box and taken up to the launder, having a draw-off for supplying any of the leaching tanks by means of an automatic distributor.

The elevator is shown on the right of Fig. 4, which, together with Fig. 5, gives a view of the leaching plant.

The leaching tanks are fitted with filters, and connections



FIG. 5.-LEACHING TANKS.

are provided so that by means of pumps the solution may be drawn from any leaching tank to either of the two sump tanks and also from either of the latter to any of the strong solution tanks or weak solution tanks. In general, the connections between the different tanks and pumps are very conveniently arranged, so as to meet all requirements of practice. Since the solution from the leaching tanks will leave a mud residue in the strong solution tanks, it is necessary to avoid drawing this off with the solution to the precipitating tanks. For this purpose a special simple but very effective means is provided.

 Λ double flange is fitted to the bottom of each of the strong solution tanks. The under half connects with the pipe line beneath the floor, going to a Montejus tank. Into the top half

of the flange a 2-inch pipe about 2 feet long is screwed. Over this 2-inch pipe a bell is allowed to hang. This bell is made of a 4-inch pipe, 2 feet long, with a cap to raise and lower it. This bell is hung over the top of the 2-inch pipe, so that its bottom edge will be at the level it is desired, to allow the mud to settle before cleaning, so that the clear solution will never be drawn off below this desired point. This clear solution is allowed to flow by gravity to the Montejus tank, which is operated in the wellknown manner by compressed air to supply the solution to the precipitating tanks.

It is to be expected that with vanadium now available in any quantities for practical purposes, it will soon cease to be considered a rare metal by steel men. The Vanadium Alloys Co. will then reap the fruit of its progressive pioneer work.

A New Slag Car.

A new and interesting type of slag car has recently been designed by the Power & Mining Machinery Co., of Cudahy, Wis. The two views shown here are of a tenton slag car, of which fifteen were recently built by the above firm for the new Garfield plant of the American Smelting & Refining Co., near Salt Lake City. Some idea as to its size can be gained from the over-all dimensions, which are 15 feet long and 6½ feet high. Each car weighs 27,000 pounds empty, or 47,000 pounds when filled with slag.

The bowl is made in five sections, the bottom being made in one piece with four quarters forming the top. This allows of any worn or broken part being easily replaced. These sections are securely bolted together and tied to a steel supporting ring



FIG. I.-SLAG CAR.

by key-bolts. The supporting ring is securely riveted to the pinion-toothed trunnions which roll on a racked track.

The pinion and the rack are both made open between the teeth to prevent their being clogged by any slag which might be spilled into them. The trunnions are rolled by means of a sleeve, which is operated through a lever by a 12-inch compressed air cylinder.

This cylinder is designed to operate on 60 pounds air pressure, and to tilt and hold the pot in any desired position. The air valve for this cylinder is a simple four-way cock, conveniently operated from the side of the car.

All the levers and other operating parts are made of steel, to reduce the weight of the machine without sacrificing strength. A hand-wheel and screw is also provided so that the bowl can be dumped by hand, should the air supply fail from any cause. When tilted to the extreme position the side of the bowl makes an angle of 50° with the vertical, so as to readily allow the slag skull to slide out. The bowl in dumping is carried to the side of the car by the rolling of the trunnions, so as to dump outside of the rail. An apron in the middle of the truck prevents any drippings from falling on the track.

The entire body of the car is carried on eight powerful springs, placed under the front and rear supporting frames

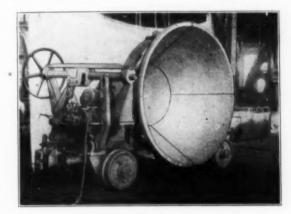


FIG. 2.—TILTING OF SLAG CAR.

and resting on the main bearing boxes. These bearing boxes are fitted with phosphor bronze bearings, which are kept lubricated by removable oil waste boxes. The two supporting frames are tied together by two 100-pound rails securely riveted to them and extended to carry the Tower automatic couplers. These are provided with the usual side-crank attachment for releasing the coupler. One of each pair of wheels is provided with a bronze bushing, so that it can revolve on the axle when going around a curve.

A large factor of safety was used in the design of all the parts of this machine, because of the rough usuage to which it is put. Special pains were also taken to make it as simple as possible to operate for the men usually employed for such labor,

Slag cars of this design are made in various sizes up to 15 tons in capacity, and with or without the air cylinder.

Enamelled Cast-Iron Apparatus for Chemical Industries.

We have received from the Mannheimer Eisengiesserei und Maschinenbau A. G., through Messrs. Frederick Bertuch & Co., New York City, their catalogue 3, containing a large number of illustrations of the numerous cast-iron apparatus made by this company for use in the chemical industries.

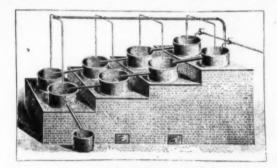
According to the special purpose for which the apparatus is intended the composition of the iron is chosen and the company has found it convenient to establish three groups, according to whether the cast iron has to resist the effect of great heat or of acid or of alkaline solutions. Certain characteristic compositions of iron answer these three purposes fairly well.

Of still greater interest, however, are the enamelled cast-iron apparatus which are made by this company in sizes up to 2.3 meters (7½ feet) external diameter and 3.2 meters (10½ feet) depth; that is any sizes up to 10,000 liters (2,600 gallons). There are three grades of enamel used, two of white and the third one of bluish-gray color. The white enamel is used

in such cases where such strong acids are to be handled which would gradually attack any bare unprotected iron, however careful its composition might be chosen. The white enamel resists satisfactorily the attack of acids, and at the same time adheres so firmly to the walls of the vessel that even considerable heat evolved by the processes is not detrimental. Care must be taken, however, to protect the enamel, which is like glass, against punching with sharp instruments. In stirring apparatus the stirrers are also enameled so as to resist the acid.

On the other hand, the blue enamel is not as well adapted to resist acids, but is extremely durable and refractory. The kind of enamel must, therefore, be chosen according to the requirements of the special case.

The adjoining illustration shows an evaporation plant for alkaline solutions made by this company. It consists of eight vessels, diameter 700 mm., depth 540 mm., content 165 liters. They are arranged as shown in the figure so that the solution flows over from the upper two vessels to the next two, and so on. The upper two receive the solution from the reservoir. The fire-hearth is placed below the two lowest vessels, so that the liquid in the same, which has already the highest concentration and, therefore, the highest boiling point, is subjected to the highest temperature. By the arrangement of tubes above the vessels, shown in the illustration, a hot-air current is produced by which hot air, which has passed a superheater, is



EVAPORATION PLANT.

blown upon the surface of the liquid in each vessel. Any fog is thereby removed, the evaporation is accelerated and foaming liquids are prevented from overflowing. The operation is continuous and very economical, since great attention is paid in the design to the thorough utilization of the fire gases.

Acetylene for the Laboratory.

By J. K. Rush.

In view of the fact that many of the best chemists in this country are not yet aware that acetylene gas is very valuable wherever an intense heat is desired, I take pleasure in throwing a little light on this subject. I was the first one in this country to install acetylene for laboratory purposes on a large or commercial scale, and I have nothing to regret for having done so, for although it was a radical departure from the uses for which acetylene was employed at that time, as well as at the present, it has given the very best of satisfaction. Some of the statements made by chemists who are using the systems which I installed, show on the face that they are made by enthusiasts, and that acetylene has fulfilled all that I had hoped it would.

Acetylene is very rich in heat units, having 1,860 heat units per cubic foot.

The exact quantity of oxygen used to produce perfect combustion varies somewhat according to the construction of the burner and the pressure of the gas at the burner. Authorities differ slightly as to the exact amount of oxygen required to produce the best results in a Bunsen flame.

By way of comparing acetylene with other gases, I will say that the best method is to ascertain how long it requires a given amount of gas to bring 1 gallon of water to the boiling point. Then compare the cost of the gases used. The following figures show the relative cost of acetylene for heating purposes as compared with coal gas and gasoline: Acetylene, 2½ cubic feet per hour, boiled in 7 minutes. Coal gas, 15½ cubic feet per hour, boiled in 8½ minutes. Gasoline consuming 1 gallon per burner every 8 hours, 7 minutes. Thus it will be seen that acetylene compares favorably with city gas at \$1.25 per thousand and gasoline at 16 cents per gallon, figuring 5 cubic feet of acetylene per pound of calcium carbide.

The heat produced by acetylene is very intense; there is a

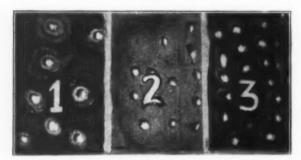


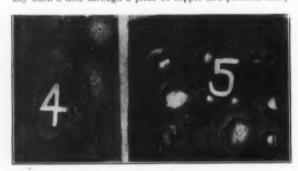
FIG. 1.-MICA.

FIG. 2.-ASBESTOS.

FIG. 3.—IRON SCREEN.

Holes in each burned with 1 second's exposure.

purplish hue with a cone of greenish-colored flame next to the orifice from which the gas is emitted. When applying acetylene with a blow-pipe with air pressure, it is possible to literally burn a hole through a piece of copper and perform many



FIGS. 4 AND 5.—OTHER EXAMPLES OF THE EFFECT OF THE ACETYLENE BLOW-PIPE.

other things which are absolutely impossible with any other gas. As a matter of fact, the intensity of the heat produced with acetylene can only be compared to that of the electric arc for intensity, but acetylene is much more convenient.

Acetylene is now being successfully used for laboratory purposes by chemists for analytical work, in some instances on a large scale. In one of the largest steel plants in the United States acetylene is now used for these purposes after other gases had formerly been used. The results have been extremely satisfactory, since acetylene was found to be far superior to either natural gas or city gas, especially in view of the fact that this intense heat can be obtained free from smoke.

With such a rich gas as this to deal with it was no easy matter for manuscuturers to design burners suitable for the acetylene Bunsen flame. But the obstacles which required constant and careful study to overcome have now been mastered to such a degree that it is possible to procure acetylene burners for practically every purpose conceivable. For the purpose of showing what an intense heat may be produced with acetylene under proper conditions and with the right kind of apparatus, I submit photos of specimens of work done with the blow-pipe, which speak for themselves. These are reproduced in Figs. 1 to 5.

One very important feature in connection with the use of

acetylene for laboratory purposes, is the fact that according to the statements made by those who are using it extensively and on a large scale, "it does not corrode nor cause any deterioration of platinum ware."

Of course, it is most important to employ the proper apparatus with acetylene. It is now generally conceded that only when it is generated from carbide at a very low temperature is it possible to get it in its purest state and in satisfactory quantities. At Cornell University, Ithaca, N. Y.,



FIG. 6.—ACETYLENE GENERATOR. DISSOLVING PROCESS.

extensive tests have been made during the past year, and are still going on, whereby it has been shown that the patent process known as the "dissolving process," produces a coolness not obtained in any other system of generating acetylene, and that it surpases all others both in quality and quantity of gas produced.

This "dissolving process" is extremely simple and unique in its workings, and is operated upon the following lines: When carbide and water come in contact with each other a large amount of heat is set free, and it is necessary to get rid of this heat. In all other systems of dropping carbide into water, a tank, or dead body of water is used, wherein only the surface of the water gets heated, as the slacking of the carbide is very rapid. Consequently the surface of the water where the acetylene is generated assumes a temperature near the boiling point, while the remainder of the water in the tank remains relatively cool.

Now, the feature of the "dissolving process" is the provision of means for radiating and circulating the entire body of water during the process of generating gas. On this account the temperature of the water is kept at from 100 per cent to 150 per cent degrees lower than with other systems, and hence the improvement in quality of the gas produced, together with a gain of from 10 per cent to 35 per cent in quantity, which is a matter worthy of consideration when the installation of an acetylene plant is being considered.

CANANDAIGUA, N. Y.

Extensions of West Allis Works of Allis-Chalmers Company.

The new extensions to the West Allis-Milwaukee works of the Allis-Chalmers Co., which, when completed, will add 861,000 square feet to the plant's present floor area of 652,000 square feet, and make the entire plant capable of affording employment to 11,000 persons, will place the West Allis works among the large industrial plants of the country. The extended works, together with the capacities of the other works of the company in Milwaukee, Chicago, Cincinnati and Scranton, will be capable of affording employment to a total of approximately 18,000 persons.

The extensions built are as follows: Three machine shops, parallel to the existing units, Nos. 1, 2 and 3, each 575 feet long and 145 feet wide for units Nos. 1 and 2 and 168 feet wide for unit No. 3; an erecting shop, 1,136 feet x 113 feet, running north and south, as an extension of the existing erecting shop and adjoining the six machine shop units opening out into it; the extension of the existing foundry, north and south, which has a total length of 994 feet over all and a width of 222 feet, and the extension of the pattern and pattern storage building, which also has a length of 994 feet over all and a width of 110 feet.

The work of constructing these extensions involves an expenditure of over \$3,000,000. Two buildings, the extensions to the erecting shop and the pattern shop, have been practically completed, while substantial progress has been made on the remaining buildings.

The West Allis site, situated in the town of West Allis, on the outskirts of Milwaukee, has a frontage of 1,575 feet, and runs back 2,696 feet, or more than half a mile, giving nearly four and a quarter million square feet of ground space, or about 100 acres, adjoining the tracks of the Chicago, Milwaukee & St. Paul Railway.

The general plan of the West Allis works provides for two sets of building units, one composed of those in which the work is common to every class of product, like the pattern shops, erecting shops and foundry; the other group, comprising the machine shop units, where the work is specialized for various machine operations. The two groups of buildings lie at right angles to each other; those of general utility lengthwise of the plant, north and south. The machine shops, six units in all, run crosswise of the plant, adjoining the erecting shop at the east end. Facing the ends of the machine shop units Nos. 1 to 6 stands the foundry with its extensions, extending north and south. The pattern storage shops stand farthest westward in the group and parallel to the foundry, with a span of 98 feet between them.

The extensions to the erecting shop, foundry and pattern shops are duplications of the existing units in practically every detail of construction. After the completion of the works we hope to give a full illustrated description.

Instrument Factory.

The rapid growth of the Leeds & Northrup Co., of Philadelphia, is indicated by the accompanying illustrations, which show the new quarters recently occupied by this firm. The building was designed and built to meet the special require-



FIG. I .- SHOP AND LABORATORY OF THE LEEDS & NORTHRUP CO.

ments of constructing and adjusting electrical measuring instruments. It has embodied in it many features of interest. The location, at 4901 Stenton Avenue, is away from the thickly settled part of the city, and has, therefore, all the advantages of ample light and freedom from dust. In selecting the site

for location, a lot of sufficient size to allow for future expansion was acquired.

The main structure is 110 feet x 52 feet, three stories in



FIG. 2.-SHOP.

height, fire-proof in construction throughout, and has all provision for the comforts of the employee.

The laboratory occupies 85 feet of the third floor, and by means of skylights in addition to numerous side windows, an abundant supply of light, so essential for work of precision,



FIG. 3 .- NORTH SIDE OF LABORATORY.

is furnished. The illustration shows the north side, which is practically a duplicate of the south side. Careful study has been given to the arrangement, distribution and insulation of the circuits for use in current, potential and general testing work. This current is furnished by a large battery of storage cells, the terminals of which are connected to a specially constructed mercury switchboard. This switchboard permits grouping the cells in any desired combination for any particular circuit. The battery room is shown at the end of the laboratory, on the right, with the charging panel outside. The mercury switchboard is not visible. All calibrating and standardizing apparatus is permanently set up in circuit, with galvanometers best adapted for the demands to be made upon them. It will be noted from the illustration that the type H galvanometer is extensively used.

In conjunction with the laboratory, but in a separate building, provision has been made for work in high temperature measurement.

The shop occupies 85 feet of the second floor, and is electrically driven. The equipment of tools and machines is specially adapted for the production of electrical measuring instruments. Co-operating with the shop and laboratory is an

inspection department, through which all apparatus must pass before shipment can be made. This inspection is most rigid, involving both mechanical and electrical tests, and thus insures the highest standard of workmanship, material and adjustment.

The basement is taken up by the heating and lighting plant, shipping room, locker room and wash room for the mechanics, fire-proof vault for records and a well-lighted room, 52 feet x

55 feet, for future manufacturing purposes.

The drafting and designing department is

The drafting and designing department is located on the second floor so as to be in close touch with the shop. In this department are filed accurate records and drawings of all apparatus manufactured. The sales office, stock room and show room are located on the third floor. In the show room are displayed, in easily accessible wall cases, a complete line of the more important instruments manufactured. In addition to this display a number of instruments are connected in circuit for demonstration under actual working conditions.

The occupancy of the new shop and laboratory has permitted



FIG. 4.-INSPECTION DEPARTMENT.

the installation of additional machinery and an increase in their mechanical and laboratory forces. As a result, a larger amount of manufactured stock and standard parts will be carried, which will insure prompt deliveries.

A cordial invitation is extended by the company to all interested persons to visit and inspect the new plant of the Leeds & Northrup Co.

Pyrometers.

A direct-reading thermoelectric pyrometer has recently been placed on the market in England by Messrs. Crompton & Co. This is another indication of the great interest which is now being paid to the importance of measuring temperatures exactly in industrial work.

There is nothing essentially new in the principle of the Crompton pyrometer. Like all thermoelectric pyrometers it consists of a thermoelectric couple, the indicating instrument

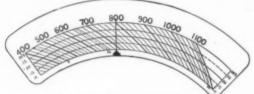
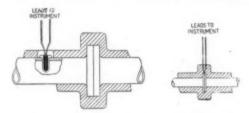


FIG. I .- SCALE OF INSTRUMENT.

and the leads connecting the couple and the instrument. For temperatures up to 1,000° or 1,100° C. the couple is constructed of nickel and steel. A nickel rod runs down the center of a steel tube without touching it, and at one end the two materials are welded together. At the other end terminal clamps

for the connecting leads are brazed to the rod and tube. For measuring temperatures of boiler flues, superheated steam, and in other cases where the temperature does not exceed 500° C. a constantan-copper couple is used. For measuring temperatures at various parts of a steam system, such as valve boxes,



FIGS. 2 AND 3 .- CONNECTION OF LEADS TO INSTRUMENT.

steam chests and different points of the pipe line, small wells or pockets are fixed at the points where the temperature is to be measured, as indicated in Figs. 1 and 2.

In cases where the steam pipes are too small to admit of a pocket, and where there is no convenient flange joint, the couple should be fastened to the pipe and heavily lagged. The lower the temperature to which the couple is continuously subjected, the longer it will last, and copper constantan couples, when measuring temperatures up to 500° C., are said to have a very long life. The life of the couple is considerably increased by protecting it with a renewable steel sheath or a porcelain cover, or by embedding it in a shallow trough containing silica. Platinum wire pyrometers resist the action of the furnace gases permanently if protected by a porcelain tube, but couples of this description are costly and easily broken.

The scale of the instrument is constructed to take into account variations in the temperature of the indicator, these variations being measured by a small thermometer attached to the face of the instrument. The several arcs are calibrated for different temperatures of the indicating instrument, and the reading should be taken on that arc which corresponds with the thermometer reading. For this purpose the end of the pointer is finished to a knife edge, which crosses radially the whole depth of the scale. Since close proximity of iron may alter the readings to some extent, the instrument should not be fixed in close contact with an iron mass, such as a pillar, but where possible it should be fixed to a wooden support or brick wall.

Oxy-Hydrogen Apparatus for Welding.

By FRANK C. PERKINS.

The accompanying diagram, Fig. 1, shows the arrangement of a new German electrolytic equipment for the production of

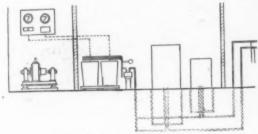


FIG. I.—PLAN OF ELECTROLYTIC PLANT.

oxygen and hydrogen gases, while Fig. 4 illustrates the method of using the oxyhydrogen flame for welding purposes. Fig. 3 shows a large plant for producing more than 40,000 cubic feet of hydrogen, and more than 20,000 cubic feet of oxygen in a day of 24 hours, and Fig. 5 a smaller cell for the electrolytic

production of oxygen and hydrogen, as constructed by the Schuckert Co., of Nürnberg, Germany.

Up to the present time the process of welding by the use of oxyhydrogen gases has been very satisfactory, but has been too expensive for very extended use, since the hydrogen and oxygen have been produced largely by chemical means. The electrolytic production of oxygen and hydrogen has been well known ever since the beginning of electrochemistry, but commercial apparatus have been placed on the market only during the last years. At the present time, in addition to the Schuckert electrolytic apparatus, an electrolyzer has been designed by the Oerlikon Co., which was described and illustrated in this journal, Vol. II., p. 293.

The Schuckert apparatus consists of a cast-iron tank, containing a number of cast-iron electrodes in various chambers separated by diaphragms, extending from the top downward about three-fourths the depth of the cell, the gases being conveyed through a pipe system to separators, whence the wash water is returned to the electrolytic cells.

The electrolyte is a 20 per cent solution of caustic potash. The cells are embedded in a sand layer about 2 or 3 inches in thickness, arranged to protect the apparatus from heat radiation, the temperature of the electrolyte being maintained at about 75° C. This is said to be the most satisfactory temperature, as the lowest voltage is required at this temperature for decomposing the electrolyte. The pressure is

particularly for that purpose. A more detailed discussion of autogenous lead soldering was given by M. U. Schoop in this journal, Vol. III., p. 260.

When current is obtained at low rates from hydroelectric power plants, or from large central power stations and rail-

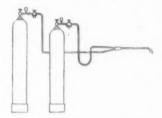


FIG. 2.—GAS TANKS CONNECTED TO BURNER.

way plants, the cost of production is low. It is stated that the time for welding I meter of sheet iron 3 mm. in thickness is about 15 minutes, while for welding I meter of sheet metal .5 mm. in thickness, it is from 4 to 6 minutes.

The Schuckert electrolyzers produce the oxygen and hydrogen gases of 97 to 99 per cent purity. The diagram of the electrolyzer plant noted in Fig. 1 shows the direct-current

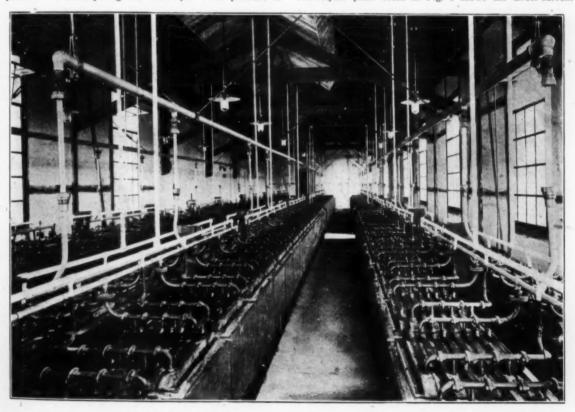


FIG. 3.—ELECTROLYTIC PLANT FOR A DAILY PRODUCTION OF MORE THAN 40,000 CUBIC FEET H AND 20,000 CUBIC FEET O.

from 2 to 3 volts, and the various cells are connected in series very much the same as a battery of accumulators. The hydrogen and the oxygen gases when generated at the electrodes are conducted through pipes to separate gasometers or tanks for storage

The use of hydrogen and oxygen is of great importance, particularly for lead burning, and is employed to advantage for welding purposes, the electrolyzer illustrated being designed generator at the left, the switchboard above it, connected with the electrolyzer in the center, the piping system conveying the gases to the gasometers at the right, the larger tank being provided for the hydrogen and the smaller for the oxygen gases.

The gases when generated by the electrolyzer are often pumped into oxygen and hydrogen tanks for storage under pressure, and these tanks are transported to any manufacturing plant where welding or lead burning is to be done and connected to the oxyhydrogen burner, as noted in the accompanying diagram, Fig. 5.

Mining and Scientific Press.

In the disaster which has befallen San Francisco, our esteemed contemporary, the *Mining and Scientific Press*—the leading mining journal of the West—has been a heavy loser. All the records, the library, the note-books of the editors, the manuscript for the printer and the whole of the plant in the

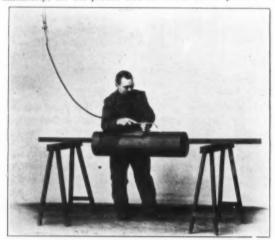


FIG. 4.-WELDING WITH OXY-HYDROGEN FLAME,

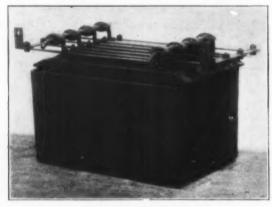


FIG. 5.—CELL FOR PRODUCTION OF HYDROGEN AND OXYGEN.

composing room lie buried under the ruins. Fortunately, a duplicate of the subscription list had been kept in Berkeley, and was thus preserved. But it is a splendid sign of the spirit which reigns to-day in San Francisco, that while the fire was still raging new offices and a new printing shop were secured in Berkeley, the new address of the journal being at the First National Bank Building at Berkeley.

In an announcement to the readers, Mr. T. A. Richard, the distinguished editor of the Mining and Scientific Press, writes as follows: "Our plant has been demolished, but this journal is built on nothing so ephemeral as paper and on nothing so cheap as machinery; it is based upon the support of many thousand readers. * * * The good-will of the Mining and Scientific Press is locked up in no safe, confined to no printing room; it cannot be shaken by an earthquake or consumed by fire. And there is another something that is not destructible by physical misfortune or financial advertising, and that is the spirit that gives life to the printed word."

Notes.

American Eelectrochemical Society.-At the meeting of the board of directors, held on April 7, the following gentlemen were elected members: E. W. Bacon, Philadelphia, Pa.; J. Lainson Wills, Brooklyn, N. Y.; Edward E. Free, Ithaca, N. Y.; B. G. Klugh, Marquette, Mich.; Philip C. Walsh, Jr., Newark, N. J. At the next meeting of the board of directors, which will be held in Ithaca on May 1, the names of the following gentlemen will come up for election: E. A. Ashcroft, New York City; Arthur Hough, Dover, N. J.; H. H. Fries, New York City; A. A. Breneman, New York City; Fanny R. M. Hitchcock, Philadelphia, Pa.; A. L. Queneau, South Bethlehem, Pa.; C. M. Joyce, Arlington, N. J.; J. E. Teeple, New York City: David Weeson, Montclair, N. J.; George Archbold, New York City; Chas. W. Moulton, Poughkeepsie, N. Y.; Irvin Kline Giles, Ithaca, N. Y.; S. H. Blake, Pittsfield, Mass., and Julius Kahn, New York City.

Metallurgy and Electrometallurgy at Lehigh.—The following is a list of the titles of seniors' theses on metallurgy and electrometallurgy at Lehigh University: W. H. Hendricks, a calorimetric investigation of copper slag and matte; H. M. Burkey and A. W. Moore, on the electric conductivity of melted metals; S. J. Cort and H. R. Lee, design and construction of an induction furnace; Y. H. Hardcastle and F. R. Pyne, an investigation of the properties of cryolite-alumina electrolytes; P. H. Herman, on the electric conductivity of some refractory materials.

Protective Coatings.—The American Society for Testing Materials, at its annual meetings, provides for a special session for the consideration of protective coatings. The executive committee has approved of the suggestion that this subject of protective coatings be amplified and made to include all paints, varnishes, etc., having protective purposes, although not designed exclusively for the protection of iron and steel. An advisory committee, consisting of G. W. Thompson, chairman, Robert Job and S. S. Voorhees, has been appointed to suggest a programme for the next meeting.

A New Departure in Electroplating.-The New York General Metal Works, with offices at 9 East Twenty-second Street, New York City, is the consoldiation of the Nonick Co., controlling patents of Mr. G. F. Marsh, and of the Architects' Bronze Works, owners of Dr. John F. Daly's patents on architects' bronze and metal lace inventions. While the chief product of the company is at present architects' bronze they also make metal grills, balustrades, newel posts, metal window frames, chandeliers and other specialties of wood or plaster, etc., and covered with an electrodeposit of bronze, copper, brass, silver or gold. The core material may be left in or taken out; in the latter case the shell of the electrodeposited metal only remains. The company also makes what they call Nonick china, a seamless and indestructible metal rim being deposited around the edge of china or glass or on any part desired. Electrodeposits are also made on lace curtains, lamp shades, etc., which process not only contributes much to the beauty of the articles but also makes them fireproof.

Division of Willson Aluminium Co.—We have received the following official announcement: "The Virginia Laboratory Co., 99 Cedar Street, New York City (Geo. O. Seward, president), has been formed for the purpose of owning and exploiting the processes and patents of F. von Kügelen and Geo. O. Seward. The Virginia Electrolytic Co., 99 Cedar Street, New York City (Geo. F. Seward, president; Geo. O. Seward, vice-president and general manager), has leased the hydroelectric plant at Holcomb's Rock, Va., from the Willson Aluminum Co., and will operate under licenses from the Virginia Laboratory Co. The Willson Aluminum Co. will continue its ferroalloy business at its plant at Kanawha Falls, W. Va." The chief product of the Willson Aluminum Co. is

ferrochrome. The extensive experimental work of Messrs. F. von Kügelen and Geo. O. Seward during the past years in Holcomb Rock had to do especially with the production of metals and alloys free from carbon and with the refining of metals and alloys in general; they have also developed an electrolytic process for winning tin from scrap. The patents relating to these inventions have been abstracted in our columns. It was reported some time ago that a detinning plant is to be erected, using the Kügelen-Seward process, but no official confirmation of this report could be obtained.

Samples of Standardized Iron and Steel.—For several years the American Foundrymen's Association has prepared and distributed samples of standardized irons. By an agreement with this Association, the samples which have been prepared under its direction have been transferred to the Bureau of Standards, and this Bureau will in the future have charge of the preparation and distribution of such samples, and will also, as soon as practicable, undertake the preparation and distribution of samples of standardized steels. At present only a single copy of steel has been prepared. Further information may be found in Circular 11 of the Bureau of Standards, Washington, D. C.

Morgan Gas Producer in Europe.—The following report from Europe is interesting, with respect to the progress made abroad by the Morgan continuous gas producer, which is so well known in this country. Messrs. Ehrhardt & Seimer, who are the European manufacturers, state that after a sharp competitive test against the Poetter producer, they have received an order for twelve 10-foot Morgan producers from a steel works in Bochum. They have also an order from the Société St. Gobain, in Pisa, for six feeding mechanisms, also an order from the Milowice Iron Works, in Russian Poland, for three producers. The latter, they think, is an entering wedge to a considerable producer business in Russia.

Gas Engines.—The National Carbon Co., of Cleveland, Ohio, have for some time operated gas engines of different makes up to 200 hp. in their various factories. They are now installing in their works at Clarksburg, W. Va., a 600-brake horse-power Koerting gas engine, supplied by the De La Vergne Machine Co., of New York. The engine will be used for general power purposes, and will operate on natural gas.

"Metals."—Mr. Sherard Cowper-Cowles, the well-known British electrochemist, and formerly proprietor and editor of the *Electrochemist and Metallurgist*, is now editing a fortnightly metallurgical section of the London *Engineering Times*. The first instalment appeared in the issue of March 22.

Steam Turbines.—The increasing demand for Westinghouse-Parsons steam turbines is indicated by the fact that the estinghouse Machine Co. received during the months of February and March orders for thirty-five steam turbines, aggregating approximately 50,000-brake horse-power capacity. The largest is of 7,500-kw. capacity, or 11,000-brake horse-power, and will be installed by the Transit Development Co., of Brooklyn. It is of the well-known multiple-expansion parallel flow type, and will drive a direct-connected alternator running at 750 r. p. m., and developing 10,000 electrical horse-power, at full load. By means of a secondary governor valve, with which all Westinghouse-Parsons turbines are provided, 12,000 kw. may be developed when running condensing, or full load when operating without condenser. This corresponds to a maximum capacity of at least 16,000 hp. at the shaft.

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Things Chemical.—Under the title "Things Chemical," a monthly bulletin is published by the Charles E. Sholes Co., of 164 Front Street, New York, in the interests of the J. T. Baker Chemical Co., the Georgetown Chemical Works, the Naugatuck Chemical Co., the Industrial Laboratories, the Triolet Co., the Metzger Chemical Co. and the Joseph Binns Chemical Works. The first number of the bulletin appeared

in April, and deals in a very interesting and amusing style with the products of the above named companies. A brief price list is also added.

Bacteriology.—The handsome and voluminous catalogue E of the Bausch & Lomb Optical Co., of Rochester, N. Y., gives an illustrated review of the various types of bacteriological apparatus made by this company, which meet all requirements of bacteriological laboratories.

Electric Hoists.—A recent neatly illustrated pamphlet of the Wellman-Seaver-Morgan Co., of Cleveland, gives descriptions of the various types of electric hoists made by this company, both for operation by direct and alternating-current motors.

Blowers and Fans.—We have received from the American Blower Co., of Detroit, Mich., a bound set of their chief catalogues, which contain a very large amount of useful practical information, with numerous illustrations. This set of catalogues is, in fact, a reference work on blowers, fans, mechanical draft, etc. The catalogues deal in detail with the following subjects: "A B C" blowers for cupolas, forges, melting and heating furnaces, forced draft, pneumatic tube systems and all other appliances where a strong, steady air pressure is applied; A B C exhaust fans, disc ventilating fans, steel plate fans, heating apparatus, fan system of heating and ventilation, mechanical draft, forced and induced by blowers and exhaust fans, two interesting papers by F. R. Still and S. S. Howell on mechanical draft; waste heat drying system, dry kilns for timber purposes, steam pumps, vertical engines.

Smelting.—Bulletin 50 of the Traylor Engineering Co., of New York City, deals with furnaces and smelting accessories, and contains many illustrations of modern types of copper matting furnaces, silver-lead furnaces, fore-hearths, copper converters, cupelling furnaces, blowers, etc.

Personal.

Dr. G. G. REVAY, of \$5 Cedar Street, New York, the wellknown fuel specialist and an authority on modern by-product coke ovens, left on April 18 for Europe, per steamship "Kaiser Wilhelm der Grosse." Several members of the staff of the United States Steel Corporation sailed on the same steamer, for the purpose of inspecting different plants in connection with the big projects now under consideration at the various works. We learn that Dr. Revay is also interested in several chemical manufactures, and intends to investigate some new electrochemical processes in Europe, which, it is said, will be established in America under most advantageous conditions, and for which purpose negotiations for large amounts of cheap water-power are carried on by some influential parties. Dr. Revay, as will be remembered, crossed to Europe last Fall in company with Mr. Charles M. Schwab and some of his engineers, to visit several important European works, and on his return worked out extensive plans for the Bethlehem Steel

Edwin Reynolds and the Design of the Horizontal Vertical Type of Steam Engine.—The award of a patent relating to the horizontal vertical type of steam engine by the United States Patent Office, Jan. 30, 1906, to Mr. Edwin Reynolds, dean of Corliss engine designers, and for many years identified with the Allis-Chalmers Co., together with the recent remarkable showing in the tests of combined vertical and horizontal engines in the New York subway power house, suggest an incident characteristic of Mr. Reynolds related by those who were close to him at the time when this type of engine was first evolved, nearly seven years ago, when the application was filed for the patents only recently granted. The incident referred to occurred at the time when the Allis-

Chalmers Co., as one of the largest builders of steam-driven units in the world, having built eleven 3,500-kw. standard cross-compound, vertical direct-connected engines for the Metropolitan Street Railway Co., was called upon for advice as to the type of engines to be used in the immense new power house then being planned by the Manhattan Railway Co., New York City. It was the intention of their engineers to install units of 5,000-kw. capacity. The first type of engine considered was the cross-compound, vertical machine, similar to those furnished for the Metropolitan Street Railway, and some correspondence passed between Mr. Reynolds and the engineers of the Manhattan Railway Company on the subject, so that finally Mr. Reynolds was invited to come to New York and discuss the matter in detail. Mr. Reynolds left Milwaukee with the question of the type of engine still unsettled, but with the understanding that the straight, cross-compound, vertical unit would be used if it were found practicable. On the train, en route for New York, however, Mr. Reynolds evolved in his mind the possibility of using a combined vertical and horizontal engine, allowing four cylinders to be connected to the one shaft, and before reaching New York he was prepared to offer to the engineers in charge a definite proposition, covering a combined vertical and horizontal four-cylinder engine, now generally known as the "Manhattan" type; and, furthermore, he submitted to them the size of cylinders which he would recommend, the size of crank-pin and cross-head pin, and the size of main journals, with rough sketches of the proposed unit as he had planned it in his mind's eye. As a result of that conversation the Allis Co. received orders for eight pairs of such machines. On Mr. Reynolds' return to Milwaukee work was at once begun on the design of the new engines in accordance with the data used by him in his New York proposition. When the complete designs were ready, it was found by comparison that in the rough notes made by Mr. Reynolds while on his way East, he had arrived at identically the same dimensions of various parts as those obtained from the carefully worked out designs. It is said that this incident is only one of a great many which have occurred during Mr. Reynolds' long service with the old E. P. Allis Co., and later with the Allis-Chalmers Co., which display his wonderful engineering ability and his promptness in making decisions. The horizontal vertical type of engine was later adopted by the Rapid Transit Subway Construction Co., showing the confidence of its engineers in these units. The patent, No. 811,520, which has now been issued to Mr. Reynolds, has been assigned by him to the Allis-Chalmers Co., who have the sole right of manufacture of this engine.

Digest of U. S. Patents.

Compiled by Barnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

Production of Ozone and Miscellaneous Gas Reactions.
(Continued.)

No. 272,187, Feb. 13, 1883, Charles E. Ball, Philadelphia, Pa. Produces illuminating or heating gas from hydrocarbon liquids, by injecting the latter into, upon or through an electric arc between carbon electrodes. It is stated that the hydrocarbon is substantially completely converted into a fixed gas, some carbon being deposited, however, upon the electrodes and serving to repair the waste of the same.

No. 282,190, July 31, 1883, Henry D. Hall, New York, N. Y. For the production of ozone, utilizes the discharge from a Holtz or other induction generator, the discharge being diffused through a space through which a current of air or gas is forced by a blower. The discharge surface may comprise wooden plates or partial partitions so arranged that the air is caused to pass between them, the partitions being provided with numerous contacting points, or metal partitions suitably

roughened or serrated may be used. In order to prevent sparking it is preferable to arrange an air gap at the rear of one or both discharge surfaces, in such manner that these surfaces are not in direct electrical connection with the terminals of the static machine.

No. 291,463, Jan. 1, 1884, Charles E. Ball, Philadelphia, Pa., and Charles S. Bradford, Jr., West Chester, Pa.

An improvement on the process of No. 272,187 above. The gas produced by vaporizing hydrocarbon liquids in arc is stated to be too rich in carbon for certain purposes, and the present invention consists in diluting the gas by inert gases, such as hydrogen. In the apparatus shown, two arcs are employed, the hydrocarbon being injected into one and water or steam into the other. The gaseous products of the two arcs are mixed and collected. The gas may be utilized for operating a gas engine, which in turn operates the dynamos for maintaining the arcs.

No. 342,548, May 25, 1886, Alfred O. Walker, Chester, England.

Utilizes high-tension discharge for effecting the deposition of fumes from smelting furnaces and for analogous purposes. In the apparatus shown, the furnace gases conveying the fumes are conducted into any suitable flue or chamber, and subjected to a high-potential discharge from metal points or equivalent projections. As illustrated, the discharge passes from a metal framework provided with points to the walls of the receptacle, one terminal of the generator being grounded. A number of similar devices may be applied at short distances along the flue. The deposit may be flushed out by water or otherwise removed in any convenient manner.

No. 361,923, April 26, 1887, Arthur Brin, 59 Brompton Crescent, County of Middlesex, England, and Leon Q. Brin, Paris, France.

An ozonizer constructed of three concentric tubes of varying sizes, disposed to leave an air-gap between the inner and intermediate tubes. The inner tube and the annular space between the outer and intermediate tubes are filled with finely-divided conductive matter, and constitute opposite discharge terminals. The divided matter may comprise plumbago, filings of iron, zinc or copper, or dust-shot. The discharge passes in the form of a large number of sparks. In the apparatus shown several ozonizers constructed as above are connected electrically in multiple, the air or oxygen to be ozonized passing through several devices in series.

No. 387,286, Aug. 7, 1888, Hartley Fewson, Buckingham, England.

Apparatus intended particularly for treating noxious gases in order to render them harmless. The device comprises a glass-lined box, provided with glass partitions so arranged as to provide a zizzag path for the gases. Each partition is faced with tin-foil on the side nearest the gas inlet, and alternate partitions are connected with opposite poles of the high-tension generator. The lead wires are of platinum, and are arranged close to but not in actual contact with the tin-foil of the respective electrodes.

No. 397,992, Feb. 19, 1889, Gottfried H. Merkel, Boston,

The ozonizing chamber, a wide, horizontal glass tube, contains a narrow glass tube which is bent back and forth into several loops. A high-resistance wire of platinum, selenium or aluminium extends through the loops and is connected at its opposite ends to an induction coil.

No. 409,903, Aug. 27, 1889, Hilarion A. B. H. de Vars, Paris.

The electrodes are a concentric inner glass cylinder lined with tin-foil and an outer glass bell-jar coated with tin-foil, both enclosed in another bell-jar. The lower ends of the cylinder and jars fit in grooves in an ebonite stand. The air passes up through the stand and cylinder and downward outside the cylinder, through slots in the stand, the ozone being delivered into a nickel-plated receiver.

